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A MINERALOGICAL EVALUATION OF THE BEHAVIOUR OF METALLIC MINERALS IN THE TROUT LAKE CONCENTRATOR CIRCUIT

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THE MINERAL CHARACTERISTICS THAT AFFECT METAL RECOVERIES FROM THE Cu, Zn, Pb AND Ag ORES FROM MANITOBA.

PART 2 : VOLUME 1.

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PREFACE

This report deals with the application of mineralogical techniques in evaluating the performance of the Trout Lake concentrator circuit. The inter-disciplinary nature of the investigation, that being the transition between the strictly academic and applied or industrial approaches to such an investigation, and the diverse interests of the readership for this work, generates it's own set of problems. Specifically, whether to report just the principal findings, or to do so with appropriate scientific justification in the form of lengthy technical discussion.

The author attempted to balance these somewhat opposing viewpoints by (A) providing considerable technical discussion throughout the body of the work, and (B) by compiling an extensive thirteen page CONCLUSION section, in which most of the conclusions are derived in brief, and are accompanied by a page reference to the corresponding section in the body of the work. The latter permits a reader not disposed to wading through text on a somewhat foreign topic, or who is interested in specific topics only, to source the relevant information directly. Thus, the CONCLUSION section may be used as a source reference, while an extended three page ABSTRACT is intended to rapidly convey the principal findings of the report.

ABSTRACT

The mineralogy of mill products from the Trout Lake concentrator circuit was investigated to determine the mineralogical characteristics affecting metal recoveries, and to evaluate and optimise process performance.

The circuit feed studied contained 2.16 % Cu, 2.57 % Zn, 2.230 ppm Au, 12.348 ppm Ag, 0.038 % Pb and 0.006 % Cd, corresponding to a Zn- and Pb-poor, Au-rich feed relative to 'normal' feeds. The recovery of Cu and Zn in their respective concentrates is 91.7 % and 82.57 %, respectively, while 3.47 % of the Cu and 11.0 % of the Zn is lost to the Zn- and Cu-concentrates, respectively. The recovery of Au and Ag is 59.2 % and 48.5 %, respectively in the Cu-concentrate, 15.3 % and 21.7 %, respectively in the Zn-concentrate, and 25.5 % and 29.8 %, respectively in the tailings. The environmentally hazardous elements As and Hg are principally lost to the tailings i.e. 79.3 % and 63.2 %, respectively. The metallurgically troublesome elements Se and Sb are principally recovered in the tailings i.e. 66.2 % and 55.4 %, respectively, whereas Sn is principally recovered in the Cu-concentrate i.e. 65.5 %. The elements are placed into five groups based on their behaviour through the circuit. These are: (1) Mn, Ni, Co and Fe report principally to the tailings; (2) As, Sb, Hg, Se and S report largely to the tailings, with moderate recoveries in the Cu- and Zn-concentrates; (3) Zn, Cd and Pb report principally to the Zn-concentrate; (4) Au, Ag, Sn and In report principally to the Cu-concentrate, with moderate recoveries in the tailings and the Zn-concentrate; and (5) Cu and Te report principally to the Cu-concentrate.

The mineralogical composition (in Vol %) of the feed is 80.37 % gangue, 9.61 % pyrite, 0.40 % pyrrhotite, 4.60 % chalcopyrite, 3.82 % sphalerite, 0.024 % galena, 0.03 % arsenopyrite, 0.00004 % freibergite, 0.00011 % electrum, 0.00004 % hessite, 0.00014 % pyrargyrite and 0.00011 % gudmundite. These minerals are placed in five groups based on their behaviour through the circuit. These are (1) the inert minerals gangue, pyrite and pyrrhotite report principally to the tailings; (2) chalcopyrite, electrum, freibergite and hessite report principally to the Cu-concentrate; (3) arsenopyrite, galena and gudmundite show low recoveries in the Cu-concentrate, with moderate to high recoveries in the Zn-concentrate and the tailings; (4) sphalerite is principally recovered in the Zn-concentrate; and (5) pyrargyrite is principally lost to the tailings, but with relatively high recoveries in the Cu-concentrate.

The simplified classification of chalcopyrite liberation in the feed is 60.6 % free, 29.8 % sub-free, 3.68 % middling and 3.27 % sub-middling grains, while 2.65 % of the chalcopyrite is unaccounted. Thus, 90.4 % of the chalcopyrite in the feed occurs in free plus sub-free or 'recoverable' classes of chalcopyrite, implying excellent 'effective liberation'; 97.3 % of which is recovered in the Cu-concentrate. The minor losses of 2.15 % of the recoverable classes of chalcopyrite to the tailings constitutes 1.94 % of the Cu in the

feed. The 1.8 % loss of Cu to the Zn-concentrate and tailings as middling chalcopyrite, negates the effectiveness of regrinding. Regrinding will address the losses of 'over-sized' free and sub-free chalcopyrite to the Zn-concentrate. Abundant oversized free and sub-free chalcopyrite and approximately 75 % of the middling chalcopyrite report to the Cu-scavenger concentrate and the Cu-1st cleaner tailings, which if combined offer the maximum potential for improved chalcopyrite liberation and Cu recovery (i.e. 1 or 2 %) through regrinding.

The simplified classification of sphalerite liberation in the feed is 50.5 % free, 41.0 % sub-free, 4.56 % middling and 1.86 % sub-middling grains, while 2.08 % of the sphalerite is unaccounted. Thus, 91.5 % occurs in free plus sub-free or 'recoverable' classes of sphalerite, implying excellent 'effective liberation'. However, 90.7 % and 9.29 % of the free, and 86.7 % and 10.3 % of the free plus sub-free grains of sphalerite report to the Zn- and Cu-concentrates, respectively. Furthermore, 96.7 % of the recoverable classes of sphalerite entering the Zn-circuit are recovered in the Zn-concentrate, indicating efficient Cu-activation by $CuSO_4$ and flotation by Minerec in the Zn-circuit. The minor loss of 3.01 % of the recoverable classes of sphalerite to the tailings constitutes only 2.75 % of the Zn in the feed. Only 2 % of the Zn is lost to the tailings and Cu-concentrate as middling sphalerite. Enhanced liberation and recovery of sphalerite by regrinding this fraction is expected to be low (i.e. 1 or 2 %).

The simplified classification of sphalerite liberation in the Cu-concentrate is 4.70 % free, 4.56 % sub-free, 0.89 % middling and 0.42 % sub-middling grains, while 0.43 % of the sphalerite is unaccounted. Thus, 85 % of the Zn in the Cu-concentrate occurs in 'recoverable' classes of sphalerite, confirming that these Zn losses are not due to poor liberation. Cu-cleaning preferentially rejects pyrite and unliberated sphalerite locked with minerals other than chalcopyrite, and concentrates free sphalerite and unliberated sphalerite locked with chalcopyrite. The Cu-concentrate is essentially 'clean', suggesting that the lack of self-draining of air bubbles in the froth, due possibly to the frothing properties of Minerec, is not the cause of the Zn losses to the Cu-concentrate.

Microprobe analyses of sphalerite indicate that Trout Lake sphalerite contains a natural range of Cu in solid solution (< 0.10 to 0.94 Wt % Cu), and that the Cu-concentrate preferentially recovers the Cu-rich species. However, the analyses failed to unequivocally characterise the Cu-content of sphalerite. Trace amounts of covellite (< 0.01 Wt %) in the feed are insufficient to cause activation of sphalerite. The covellite is derived from Cu-Fe-sulfides formed by quenching of Cu-matte entrained in slag, and which is introduced into the feed by dilution from backfilled slag. This introduction of slag and associated covellite is strongly discouraged.

The grain size distributions of sphalerite in the feed (80 % - 36 um) and Cu-concentrate (80 % - 19 um), show that the sphalerite recovered in the Cu-concentrate is preferentially finer-grained. However, the slime fraction of free and unliberated sphalerite in the Cu-concentrate is 13.5 and 9.5 %, respectively, arguing against the entrainment of sphalerite slimes as a significant cause of these Zn losses. Of the unliberated sphalerite in the Cu-concentrate 88 % is locked with chalcopyrite, and 76 % occurs in the sub-free

classes. This suggests that sphalerite is marginally floatable using Minerec collector, and that subordinate locked chalcopyrite may exert sufficient influence to carry the sphalerite grain above the threshold for flotation. Regrinding of sub-free sphalerite locked with chalcopyrite may significantly improve sphalerite rejection. The proposed regrind stream of the combined Cu-scavenger concentrate and the Cu-1st cleaner tailings, contains 24.2 % of the Zn in the feed, of which 59 % occurs as sub-free and 9 % as middling sphalerite. These proportionally large fractions offer the greatest potential for a regrind to enhance liberation and recovery of sphalerite in the Zn-concentrate. The present regrind produces a 3 % decrease in the particle size of the Cu-rougher tailings, and negligible improvement in liberation.

The persistence of diablastic-textured particles of chalcopyrite-sphalerite suggests that liberation from the Sheared Chalcopyrite + Sphalerite ores is low. As this ore type dominates the No. 2 lens of the north zone, feeds rich in ore from this lens are likely to yield poor chalcopyrite-sphalerite liberation, with significant losses to the opposing concentrates. Chalcopyrite-disease is largely liberated, occurring in approximately 5 % of the sphalerite in the feed, and comprising approximately 20 % of the unliberated sphalerite in the Cu-concentrate. Nonetheless, the bulk of the persistent chalcopyrite-disease reports to the Zn-concentrate.

The liberation of galena is 7.69 %, while this free galena 'seems' to be largely, but not wholely, recovered in the Cu-concentrate. Unliberated galena grains report to products in accordance with the flotation characteristics of the accompanying locked mineral i.e. 41.8 % of the unliberated galena reports to the Zn-concentrate principally as sphalerite-galena particles, and 47.0 % to the tailings principally as pyrite-galena particles. Minerec achieves relatively poor flotation of nominally recoverable grains of galena.

Mercurian electrum contains approximately 85 to 90 % of the Au in the ore. The recovery of electrum in the Cu-concentrate is calculated as 90 %. No electrum was observed in the tailings, but the calculated losses to the tailings are 7 %, probably as un-observed type 1 electrum. Electrum is readily recovered using Minerec collector, but with high circulating loads during Cu-cleaning. Approximately 13 % of the Au is lost to the tailings as 'invisible' Au (most probably in solid solution) in pyrite and arsenopyrite. It is inferred that pyrite is arsenian, supporting the inference of 'invisible' Au in pyrite.

It is inferred that chalcopyrite, sphalerite, pyrite and galena contain 45, 55, 11 and 750 ppm Ag, respectively, which corresponds to 23 %, 19 %, 18 % and 2.7 % of the Ag in the feed. Electrum, pyrargyrite, freibergite and hessite account for 18 %, 13 %, 6 % and 5 %, respectively of the Ag in the feed. As sphalerite is a major Ag-carrier, improvement in Zn recovery will result in routine losses of 20 % of the Ag to the Zn-concentrate. As 14 % of the Ag in the ore occurs in pyrite, 88.8 % of which is recovered in the tailings, approximately 12 % of the Ag in the ore is expected to be lost, and accounts for 40 to 50 % of the Ag in the tailings. The high recoveries of chalcopyrite, electrum, freibergite and hessite account for 86 % of the Ag in the Cu-concentrate, and corresponds to 40 to 45 % of the Ag in the ore.

CONTENTS

ACKNOWLEDGEMENTS ii

PREFACE iii

ABSTRACT iv

Chapter page

I. INTRODUCTION 1

II. SAMPLING CAMPAIGN 2

III. METHODS OF INVESTIGATION 6

CHEMICAL ANALYSIS 6
SIEVE SIZE ANALYSIS 6
PETROGRAPHIC AND IMAGE ANALYSIS 6
MATERIAL BALANCE CALCULATIONS 9

IV. THE ORE COMPOSITION OF THE FEED 11

V. CHEMICAL ANALYSIS 16

CONCENTRATION FACTORS 26

VI. MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES 31

CONCENTRATION FACTORS 39
FLOTATION EFFICIENCY 45

VII. MINERAL LIBERATIONS 51

FREE GRAINS 51
CHALCOPYRITE 61
SPHALERITE 61
GALENA 62
UNLIBERATED GRAINS 62
CHALCOPYRITE 68
SPHALERITE 81
GALENA 99

VIII. SIZE ANALYSIS 111

SCREEN SIZE ANALYSIS OF THE ROD MILL FEED 111
SCREEN SIZE ANALYSIS OF THE MILL PRODUCTS 114

GRAIN SIZE ANALYSIS OF MINERALS BY IMAGE ANALYSIS 126
GRAIN SIZE DISTRIBUTIONS OF THE PRINCIPAL MINERALS IN THE FEED 126
GRAIN SIZE DISTIBUTIONS OF FREE, UNLIBERATED AND SUB-FREE CHALCOPYRITE, SPHALERITE AND GALENA 132
CHALCOPYRITE 133
SPHALERITE 135
GALENA 147

IX. POTENTIAL MINERALOGICAL INFLUENCES ON METAL RECOVERIES 163

TEXTURAL CHARACTERISTICS OF UNLIBERATED SPHALERITE IN THE Cu-CONCENTRATE 163
THE MINERAL CHEMISTRY OF SPHALERITE 167
CONTAMINATION OF MICROPROBE ANALYSIS : THEORY IN BRIEF . 167
RESULTS 168
POTENTIAL ACTIVATORS OF SPHALERITE 171
MARCASITE 171
COVELLITE 172
THE BEHAVIOUR OF PYRITE IN THE Cu-CIRCUIT 180
MINERAL-METAL ACCOUNTING 185
SILVER MINERAL-METAL ACCOUNTING 186
GOLD MINERAL-METAL ACCOUNTING 190
ARSENIC MINERAL-METAL ACCOUNTING 195
LIBERATION OF THE Ag-MINERALS FREIBERGITE, PYRARGYRITE, HESSITE AND ELECTRUM 200

X. CONCLUSIONS 204

XI. RECOMMENDATIONS 217

BIBLIOGRAPHY 219

Appendix page

A. 222

B. 230

C. 245

D. 254

E. 261

F. 266

Chapter I

INTRODUCTION

A mineralogical study was conducted on the ores from the Trout Lake deposit, and mill products from the Trout Lake concentrator circuit as a project within the CANADA-MANITOBA MINERAL DEVELOPMENT AGREEMENT. This project entitled "The mineralogical characteristics that affect metal recoveries from the Cu, Zn, Pb and Ag ores from Manitoba" was conducted as a contract to Hudson Bay Mining and Smelting Co. Ltd. (HBM&S) from CANMET, Dept. of Energy, Mines and Resources, under the supervision and guidance of the CANMET Scientific Authority Dr. W. Petruk. The work was performed by the author at the University of Manitoba as an employee of HBM&S under this contract.

The Trout Lake deposit is a volcanogenic massive sulphide deposit that contains the recoverable metals Cu, Zn, Au, Ag, Pb, Cd, Co and Se. The objective of the project was to establish whether the volcanogenic massive sulphide deposits of Manitoba contain unique mineralogical characteritics, and to determine whether these characteristics can be utilized to modify metallurgical practises, so that recovery could be optimised. The Trout Lake deposit had been selected for study because the copper concentrate commonly contains 5 Wt % zinc, and because approximately 30 % of the gold and of the silver are lost to the tailings.

The study was divided into two parts. Part 1, to define the mineralogical characteristics that could have a bearing on mining and mineral processing, and Part 2, a mineralogical study of the behaviour of the minerals in the concentrator. Part 1 involved collecting a representative suite of samples from the deposit, fitting them into a classification based largely on that developed by the mine geologist, and characterising the ore from each classification. The ore characterisation involved identifying the minerals, determining mineral quantities, determining the average chemical composition, defining the host mineral(s) for each element of interest, determining the size distribution of the minerals in the unbroken ore, and noting mineral textures that might have a bearing on mineral beneficiation. A separate report covering Part 1 of the study has been completed (Healy & Petruk 1988).

Part 2 of the project involved sampling the Trout Lake concentrator circuit, and performing sieve size, chemical, petrographic and image analysis on each of the products sampled. The image analysis included modal analysis, grain size analysis and mineral liberation analysis. Material balance calculations were performed on a wide variety of the analytical data thus gathered, to determine the behaviour and inter-relationships of the metals and the minerals through the circuit. This paper covers Part 2 of the study.

Chapter II

SAMPLING CAMPAIGN

A sampling campaign of the Trout Lake concentrator circuit was conducted on Febuary 5th 1986. The strategy employed was to collect samples of the feed and all the discharge products (concentrates and tailings) from each bank of flotation cells in the entire cicuit (Petruk & Klymowsky 1977 and Petruk & Schnarr 1979). Due to poor access, samples of (1) the combined Cu rougher feed, (2) the Cu scavenger feed after regrinding, and (3) the combined Zn 1st cleaner feed were not collected (See Fig 1). Conversely, for four of the products, several samples of essentially the same product were collected (e.g. 5 samples of the Zn rougher concentrate) because of multiple banks or streams. Thus, the 25 samples represented essentially 17 products. Using a sampling team of 5 personnel 25 samples were collected over a period of 2 hours. A cut of approximately 500 ml of slurry was taken at each sample point every 10 minutes to produce approximately 6 litres of slurry per sample. The slurry was transferred from the sample pails to a pressure filter and the filtered solids were oven dried. The dried samples were then quartered into subsamples, bagged and labelled for chemical analysis (1 quarter), screen size analysis (2 quarters), and for preparation as polished sections (1 quarter). The samples were numbered such that the overflows and underflows were assigned even and odd numbers, respectively, and the numerical order approximated to their position through the circuit. The sample numbers are prefixed with 'TL.MP', denoting 'Trout Lake Mill Product'.

The on-stream x-ray analyser indicated that steady-state conditions prevailed during the sampling campaign, despite a large change in grades prior to commencing sampling (See Table 1). The on-stream analyser also indicated that the grade of the feed was slightly higher in Cu and lower in Zn relative to the average yearly mill head grade (See Table 2). In addition, a 10 kg grab sample of the - 1.25 inch rod mill feed was collected 'off the belt' midway through the sampling campaign. The location and stope numbers from which the ore was mined at the time of the sampling, was also recorded.

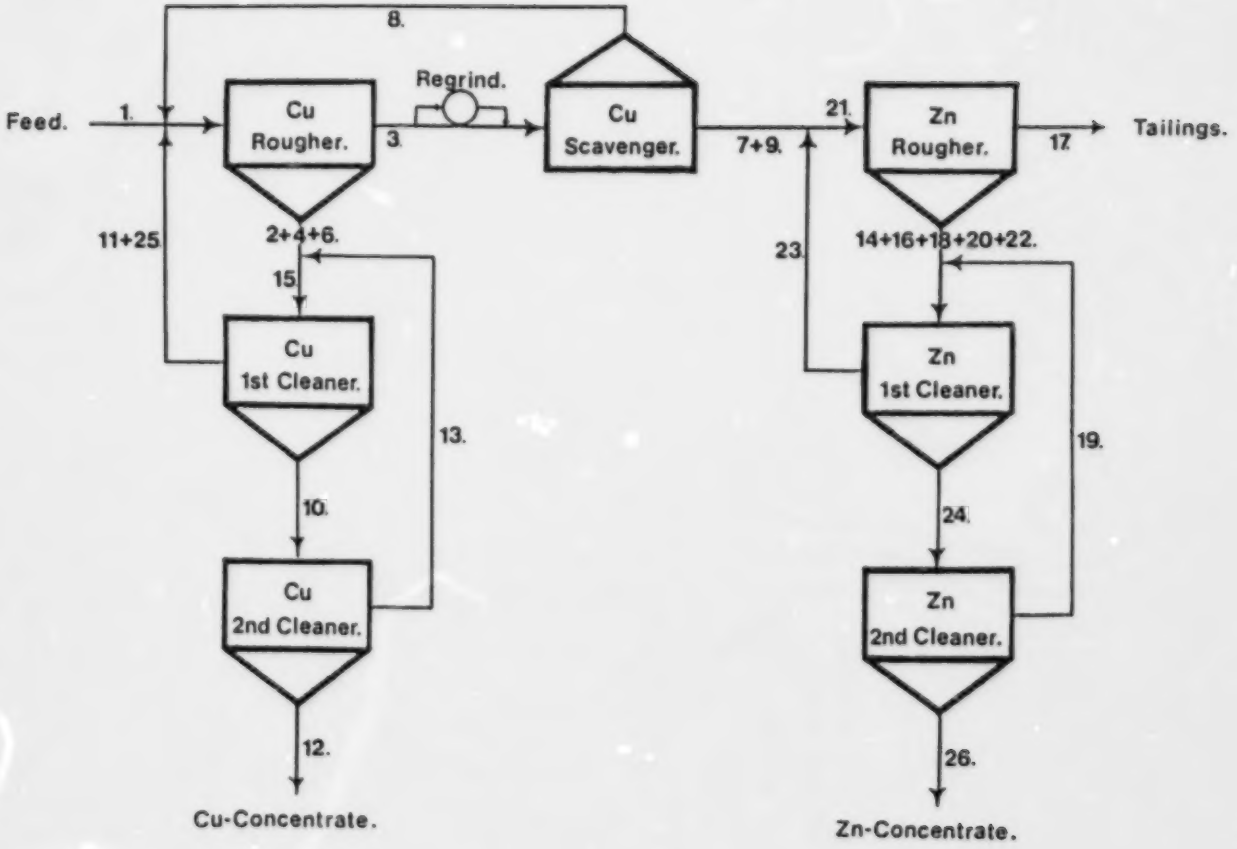

FIGURE 1.
THE TROUT LAKE CONCENTRATOR CIRCUIT PORTRAYING THE 25 SAMPLE POINTS.

TABLE 1

STREAM ANALYSES BY ON-STREAM X-RAY ANALYSER DURING THE SAMPLING CAMPAIGN.

Sampled between 9.30 and 11.30 A.M. February 5th 1986.

TIME	HEADS		Cu-CONCENTRATE				Zn-CONCENTRATE				TAILINGS	
	Cu	Zn	Cu	Cu-Rec	Zn	Zn-Rec	Cu	Cu-Rec	Zn	Zn-Rec	Cu	Zn
9.15	3.77	2.20	23.46	96.0	4.38	30.7	1.29	0.9	46.83	55.9	3.1	13.4
9.44	2.68	2.86	23.82	94.8	4.47	16.7	1.49	2.6	46.60	75.0	2.6	8.4
9.58	2.74	2.92	24.04	95.2	4.48	16.7	1.31	2.2	47.62	75.7	2.6	7.6
10.13	2.76	2.80	24.29	95.3	4.47	17.3	1.37	2.2	47.34	75.1	2.5	7.6
10.42	2.67	2.98	24.15	94.8	4.60	16.1	1.33	2.5	46.45	76.8	2.7	7.1
10.56	2.69	2.77	24.40	95.4	4.65	17.6	1.34	2.2	45.89	74.3	2.4	8.0
11.11	2.74	2.68	24.24	95.9	4.62	18.6	1.24	1.9	46.80	73.6	2.2	7.8
11.26	2.90	2.75	24.25	96.1	4.58	19.1	1.21	1.8	48.03	74.1	2.2	6.8

TABLE 2

TROUT LAKE MILL HEAD GRADES.

	Cu(%).	Zn(%).
1984 (Yearly).	2.45	3.76
1986 (Feb - This Study).		
On-Stream X-ray Analyser.	2.74	2.82
Bulk Chemical Analyses.	2.19	2.70

Chapter III

METHODS OF INVESTIGATION

CHEMICAL ANALYSIS

The twenty five samples were analysed chemically for Cu, Zn, Au, Ag, Cd, Pb, Fe, S, Co, Ni, Mn, and Bi at the HBM&S assay laboratory, Flin Flon, and for As, Se, Te, Hg, Sn, Sb and In by Bondar-Clegg, Ottawa. The dried samples required no preparation for chemical analysis, as they are fine-grained and homogeneous. The HBM&S assay laboratory has previously achieved a high degree of accuracy and precision (Healy & Petruk 1988), hence no reference ore standards were included with this batch of samples.

The sample rejects from the HBM&S assay laboratory were forwarded to Bondar-Clegg. The CANMET reference ore standards CZN-1 and MP1-A, were included and labelled as additional unknowns. Bondar-Clegg performed duplicate analyses of several submitted samples, and these indicated that good analytical precision was achieved. The accuracy of the analyses was evaluated by comparing the measured with the recommended or non-certified values quoted for CZN-1 and MP1-A. In general good agreement was obtained (See Table 3).

SIEVE SIZE ANALYSIS

The twenty five samples were submitted to HBMS for sieve size analysis. The samples were machine sieved, using stacked Tyler Series meshes with the mesh numbers 48, 65, 100, 150, 200 and 325. The weight of retained material in each sieve, and that passing 325 mesh was determined and recorded for each sample.

PETROGRAPHIC AND IMAGE ANALYSIS

The twenty five samples were prepared as polished sections at CANMET, Ottawa. Approximately 200 mg of the sample was mounted in epoxy/araldite in such a manner as to minimise fractionation of the sample by gravity settling of the particles. The polished sections were petrographically examined to qualitatively determine the abundance, median grain size, degree of liberation, and the type of locked species of the minerals of interest. This information was useful in selecting and writing appropriate image analysis software routines.

The image analysis was performed at CANMET, Ottawa on the MP-SEM-IPS-IAS image analysis system, which was developed by Petruk (Petruk 1987). The SEM

TABLE 3A

TEST OF ANALYTICAL PRECISION BY DUPLICATE ANALYSIS.

SAMPLE.	As	Hg	Te	Se	Sn	Sb	In
	Wt %	ppm					
TL.MP-9	0.07	18.	nd.				nd.
DUPLICATE	na.	18.	nd.				nd.
TL.MP-10				173.	56.	89.	
DUPLICATE				175.	62.	98.	
TL.MP-17	0.06	15.	nd.				nd.
DUPLICATE	na.	15.	nd.				nd.
TL.MP-18				85.	33.	185.	
DUPLICATE				93.	33.	173.	

TABLE 3B

TEST OF ACCURACY BASED ON ANALYSES OF REFERENCE ORE STANDARDS CZN-1 AND MP1-A.

ELEM	CZN-1 ANALYSIS	CZN-1 RECOMMENDED/CERTIFIED VALUE.	MP1-A ANALYSIS	MP1-A RECOMMENDED/CERTIFIED VALUE.
As	0.03 Wt%.	0.026 ± 0.002 Wt%.	0.84 Wt%.	0.84 ± 0.02 Wt%.
Se	27 ppm.	5 (Non-Certified).	31 ppm.	- (Not Quoted).
Te	nd.	- (Not Quoted).	nd.	- (Not Quoted).
Hg	49 ppm.	43 ± 4 ppm.	5 ppm.	- (Not Quoted).
Sn	53 ppm.	74 (Non-Certified).	>1.25 Wt %	1.28 ± 0.04 Wt%.
Sb	505 ppm.	520 ± 30 ppm.	nd.	- (Not Quoted).
In	84 ppm.	86 (Non-Certified).	337 ppm.	330 ± 10 ppm.

'nd.' denotes not detected.
'na.' denotes not analysed for.

capabilities of the Jeol 733 electron microprobe, the image producing component of the system, permits the generation of high resolution backscattered (BSE) images of the polished sample surface. The BSE (or grey

level) image of a single field of the sample surface is tranferred to the Kontron SEM-IPS image analyser and digitised. Those areas of the image detected within the grey level window of each mineral are transformed into separate binary images. These mineral-specific binary images are stored in the image memory assigned to each mineral or mineral group i.e. gangue[1] - silicates, carbonates, etc. At this point the various stereological measurements are made, and the procedure is repeated for successive fields. For further details of the specific technique used, see Healy & Petruk (1987).

Unliberated grains of the mineral(s) of interest are described on the basis of their grain size distribution and classification. The classification[2] describes the degree to which each particle containing an unliberated grain of the mineral of interest is locked with other minerals. Each grain having a common boundary with another or several other minerals is classified on the basis of the unliberated grains contribution in terms of area % to the total area of the particle. Ideally, each particle is thus described on the basis of both the area % and the size of the contained valuable mineral grain.

Polished sections of each product were analysed twice; first by a routine dedicated to the major minerals, and secondly, by a routine dedicated to the minor minerals. The major minerals routine determined the following: (1) quantities of each mineral (or mineral group); (2) liberations (% free) of chalcopyrite, sphalerite, and galena; (3) grain size distribution for the free chalcopyrite, sphalerite and galena; and (4) classification of unliberated chalcopyrite, sphalerite and galena. The minor minerals routine was employed to identify and measure similar stereological properties of electrum, freibergite, hessite, pyrargyrite and other minerals of interest. All these minerals are generally very fine-grained, present in trace amounts, and are difficult to identify because they cannot be differentiated from each other in

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[1] Gangue is defined here as non-sulfide minerals characteristic of the surrounding hostrocks, and of no economic value i.e. silicates, carbonates and phosphates.

[2] All observed grains of the mineral of interest, and which comprise a variable volume % of the particle which they are identified as occurring in, are classified as being free (100 volume % of the particle) or unliberated (< 100 volume %). The unliberated grains are in turn classified into a ten-fold classification (i.e. ten intervals of 10 volume % each) based on the unliberated grains contribution in terms of volume % to the total volume of the particle. This classification may be simplified into a three-fold classification based on the expected flotation properties of the particle. The 'sub-free' particles are those in which the grain of the mineral of interest comprises between 99.99 % and 70 % of the particle, and which is expected to behave essentially as if free, i.e. in accordance with the flotation characteristics of the mineral of interest. The 'middling' particles are those in which the grain of the mineral of interest comprises 69.99 % to 30 % of the particle, and whose flotation properties are likely to mimic that of either the mineral of interest or the accompanying mineral(s). The 'sub-middling' particles are those in which the grain of the mineral of interest comprises less than 29.99 % of the particle, and whose flotation properties are expected to correspond to that of the dominant accompanying mineral(s).

the BSE image. These were identified by scanning particles that belong to this 'family' of minerals with the electron beam and determining their chemical composition by EDXA. The minerals identified on the basis of chemical composition were analyzed to determine the following: (1) quantities of the minor minerals of interest; (2) liberations of each of the minerals of interest; (3) grain size distribution of free galena; and (4) grain size distribution and classification for unliberated galena, pyrargyrite, freibergite, hessite and electrum. Much of the data generated by both routines was summarised in tables, histograms and 3-D diagrams, and are compiled in Volume 2, some of which are presented in this volume.

Image analysis using the major mineral routine was performed on each of the polished sections corresponding to the 25 samples, whereas the minor mineral routine was not employed on all 25 samples, but on a sufficient number of samples to include all 17 principal mill products. The major mineral routine employed a magnification of x150 (pixel size = 0.936 um) over an analytical area of approximately 40 fields per sample. However, the analyses for galena and freibergite generated by the major mineral routine were rejected, because the small number of fields analysed was insufficient to provide a statistically valid number of measurements for these trace minerals. The minor mineral routine employed a magnification of x400 (pixel size = 0.368 um) over an analytical area of between 200 and 5,000 fields. An analysis of approximately 3,000 to 5,000 fields involved 2 overnight runs, and was required for those samples where the minerals of interest are so rare that possibly only one grain occurs in the plane of the section, i.e. gudmundite in the sample of the tailings.

MATERIAL BALANCE CALCULATIONS

In order to evaluate plant performance and accurately describe the steady-state operation of an ore processing circuit using quantitative measurements of selected parameters (e.g. chemical assays, or mineral liberations) from different streams in the circuit, it is necessary to perform material balance calculations (Laguitton & Wilson 1979). Because the weight of the products are unknown, and the raw analytical data contains sampling, sample preparation and analytical errors, material balance calculations are used to generate a new set of coherent data, which may include otherwise inaccessible or experimentally undetermined data. The initial material balance calculations were performed by J. Wilson and W. Petruk at CANMET, using the MATBAL computer program (Laguitton & Wilson 1979). Subsequently, material balances were performed once the MATBAL program was fully implemented on the WATFIV compiler on the University of Manitoba mainframe. The program employs the principle of mass balance, in which the feed to each bank of flotation cells is split into a concentrate/overflow and tailings/underflow, and the weight and assays (or any other measured parameter) must balance (Petruk & Schnarr 1981). Because of analytical errors in the raw data, the program uses a 'least squares refinement' technique to adjust the raw data, and produces 'best fit' or 'balanced' values, upon which most of the evaluations in this report are based.

Initially the weight of each product was determined with the MATBAL program using the chemical assays for Au, Ag, Cu, Zn, Fe and S. Having determined the weights of each product the assays were balanced by MATBAL to yield adjusted assays for all 18 elements. The MATBAL program simultaneously calculates the element recoveries in each product. Material balances were also performed on several other types of data using the previously determined weight of each product, in order to determine the recoveries of specific types and classes of mineralogical species. These other types of data are : (1) sieve size data; (2) mineral quantities (modal) data for all the minerals of interest (detected minerals); (3) mineral-metal data for liberated chalcopyrite, sphalerite and galena; (4) mineral-metal data for each class of unliberated chalcopyrite, sphalerite and galena; and (5) mineral-metal data for composite classes of liberated plus unliberated chalcopyrite, sphalerite and galena. The mineral-metal data is the 'product of the metal assay times the percent liberation (or type and/or class(es) of liberation) of the host mineral' in that mill product.

The material balance calculations were also used to determine the relative proportions of the ore types that comprised the feed during the sampling campaign. The average composition of each of the 9 ore types that are recognised at Trout Lake is known (Healy & Petruk 1988). Together with wallrock, which was assigned a 9.6 Wt % Fe assay to accomodate for silicate Fe, the 9 ore types contribute in proportion to their abundance, to the chemical composition of the feed. Thus, knowing the composition and the weight (100 %) of the feed and the composition of each product (ore type), it was possible to calculate by material balance, the weight of each ore type.

Chapter IV

THE ORE COMPOSITION OF THE FEED

The ore being processed at the time of the sampling campaign on Febuary 5th was mined from the following stopes

Stope 225N.	Lens 2.	North Zone.	255m Level.
Stope 2411.	Lens 2.	South Zone.	200m Level.
Stope 1511.	Lens 1.	North Zone.	100m Level.

The composition of this float feed, the 1984 float feed, and unbalanced assays of the 1983 float feed are presented below in Table 4. It is apparent that the feed on Febuary 5th contains average Ag and Cu contents, low Zn, Fe and S and possibly Pb contents, and high Au contents relative to 1983 and 1984 feeds. The low Zn and Fe contents suggest the feed is depleted in the Massive Sphalerite, Banded Pyrite + Sphalerite and the Massive Pyrite ore types

TABLE 4

CHEMICAL COMPOSITION OF TROUT LAKE FLOAT FEEDS.

Feed.	Au	Ag	Cu	Zn	Pb	Fe	S
	ppm.		Wt %.				
This Study.	2.195	12.01	2.19	2.70	0.037	16.1	9.7
1984.	1.441	12.14	2.45	3.76	-----	19.3	---
1983.	1.509	12.69	1.78	4.64	0.059	21.7	14.9

relative to normal feeds.

The chemical assays for 7 elements and the modal analyses for pyrite, gangue and pyrrhotite for the feed, the 9 ore types and the wallrock were used

in the material balance calculations to determine the proportions of the ore types comprising the feed. This input data are presented in Table 5. The weight of each product (ore type), and the percentage of the metals of interest in the feed contributed by each ore type were calculated by MATBAL, and are presented in Table 6.

It must first be stressed that the chemical and modal compositions of the ore types are averages based on a small number of samples (i.e. 1 to 5 samples per ore type), and are inherently uncertain (Healy & Petruk 1988). In addition, many of the ore types are mutually transitional, while others e.g. the Sheared Chalopyrite + Sphalerite ores are defined largely on textural criteria and not on the basis of chemical or mineralogical composition. Therefore, the material balance calculations are considered an approximation of the relative proportions of the ore types comprising the feed, and no

TABLE 5

THE AVERAGE CHEMICAL AND MODAL COMPOSITIONS OF THE ORE TYPES.

Product.	Au	Ag	Cu	Zn	Fe	S	Pb	Py	Gang	Po
	ppm		Wt %					Vol %		
Feed.	2.2	12.0	2.2	2.7	16.1	9.7	0.037	9.2	84.0	0.33
M. Sp.	1.5	131.0	1.7	37.9	11.6	25.3	3.100	1.9	23.4	2.70
B. Py+Sp.	3.5	86.0	0.5	16.2	27.2	34.5	1.000	51.1	17.0	0.01
M. Py.	1.2	9.1	0.3	4.3	39.2	40.6	0.008	79.7	16.2	0.02
M/B. Cp+Sp.	4.0	58.0	5.4	17.8	21.1	25.1	0.007	14.0	32.2	7.70
Sh. Cp+Sp.	6.9	65.0	5.2	23.2	17.0	22.8	0.120	5.1	29.4	11.00
M. Cp+Po.	3.4	39.0	12.1	0.4	37.3	29.0	0.004	19.5	22.5	21.30
Cp Stgs.	2.3	38.0	12.4	2.7	27.5	15.7	0.040	8.9	44.3	5.90
D. Cp+Po.	1.0	6.3	1.3	4.8	16.5	24.0	0.001	32.1	62.9	0.03
V. Qtz+Cp.	6.2	1.3	3.2	0.4	5.6	3.9	0.005	3.3	86.4	0.90
Wallrock.	0.0	0.0	0.0	0.0	9.6	0.0	0.000	0.0	100.0	0.00

Note : Data from Healy & Petruk (1988).

quantitative conclusions are put forward on the strength of them.

The most striking result is that the Chalcopyrite Stringers is calculated as having contributed 41 % of the Au, 46 % of the Ag, 84 % of the Cu, 16 % of the Zn, and 15 % of the Pb in the feed. This is significant because this ore

TABLE 6

RESULTS OF MATERIAL BALANCE OF ORE TYPES COMPRISING THE FEED.

Product.	Weight.	Au	Ag	Cu	Zn	Pb	Fe
		Percent of Metal in Feed contributed by each Ore Type.*					
M. Sp.	0.30	0.53	3.36	0.24	4.85	24.18	0.21
B. Py+Sp.	2.22	9.39	16.46	0.53	15.59	57.53	3.68
M. Py.	11.37	16.80	8.92	1.62	21.38	2.40	27.13
M/B. Cp+Sp.	2.65	12.92	13.24	6.78	20.57	0.49	3.41
Sh. Cp+Sp.	1.90	16.11	10.65	4.67	19.22	0.60	1.97
M. Cp+Po.	0.28	1.14	0.94	1.61	0.05	0.03	0.64
Cp Stgs.	13.86	41.18	45.89	83.76	16.27	14.56	23.22
D. Cp+Po.	1.00	1.19	0.54	0.62	2.05	0.03	1.01
V. Qtz+Cp.	0.10	0.74	0.01	0.15	0.02	0.01	0.03
Wallrock.	66.34	0.00	0.00	0.00	0.00	0.18	38.69

* Calculated as recovery in a product.

type is generally coarse-grained, particularly with respect to chalcopyrite and electrum, is mineralogically relatively 'simple', and because the predominant gangue mineral is chlorite, is interpreted to be readily grindable. It is also calculated that the Massive Pyrite contributed 21 %, the Mixed/Banded Chalcopyrite + Sphalerite 21 %, the Sheared Chalcopyrite + Sphalerite 19 %, the Chalcopyrite Stringer 16 %, and the Banded Pyrite + Sphalerite 16 % of the Zn in the mill feed, while the Massive Sphalerite contributed only 5 % of the Zn. The Banded Pyrite + Sphalerite, the Massive Sphalerite and the Chalcopyrite Stringer ore types contributed 57 %, 24 % and 15 % of the Pb in the feed, respectively. The principal contributors of the Au were the Chalcopyrite Stringer (dominant), the Massive Pyrite, the Sheared Chalcopyrite + Sphalerite, the Mixed/Banded Chalcopyrite + Sphalerite and the Banded Pyrite + Sphalerite. The principal contributors of Ag were the Chalcopyrite Stringer (dominant), the Banded Pyrite + Sphalerite, the Mixed/Banded Chalcopyrite + Sphalerite, the Sheared Chalcopyrite + Sphalerite and the Massive Pyrite. The Chalcopyrite Stringer ore contributed 46 % of the Ag, yet is not known to contain freibergite and pyrargyrite, nor appreciable concentrations of sphalerite and galena, which are interpreted as the four principal Ag-carriers in the deposit (Healy & Petruk 1988). It is inferred that chalcopyrite, hessite, sphalerite, galena-clausthalite and electrum are the principal Ag-carriers in the Chalcopyrite Stringer ores.

The calculated weights of the products (ore types) indicate that the principal ore types in the feed are the Chalcopyrite Stringer and the Massive Pyrite ore types, while the Mixed/Banded Chalcopyrite + Sphalerite, the Banded Pyrite + Sphalerite and the Sheared Chalcopyrite + Sphalerite are the only other significantly abundant ore types. The principal product is calculated to be the wallrock, comprising 66 % of the feed. This high percentage of wallrock in the feed is interpreted to reflect (1) interbedded barren wallrock within the ore lenses, (2) dilution from wallrock within the mining width, but not within the ore lenses, and (3) gangue occurring disseminated in the ore types, and not incorporated in to the average composition of the ore type because of a bias in the sampling towards sulfide-rich, gangue-poor samples (Healy & Petruk 1988). Wallrock consisting of the interbedded type and the dilution type is likely to occur in the ground feed as non-sulfide bearing, barren waste particles. The magnitude of the calculated wallrock component prompted an examination of the 10 kg grab sample of the rod mill feed, in order to determine the magnitude and character of the wallrock component, and also to confirm the principal ore types present.

The grab sample, consisting of less than 'one and a quarter inch particles', was hand sorted using visual discrimination. Particles less than 3 mm were too small to be sorted and were rejected. The results of the sorting are assumed to be true for the small (<5%) rejected fine-grained fraction. The Disseminated Pyrite + Chalcopyrite ore type was not picked out and probably contributed largely to the Chalcopyrite Stringer ore type and to a lesser extent to the Massive Pyrite ore type. The Sheared Chalcopyrite + Sphalerite ore type is difficult to identify in hand-specimen less than one and a quarter inches in diameter and was not sorted separately, and probably contributed to the Mixed/Banded Chalcopyrite + Sphalerite and the Chalcopyrite Stringer ore types. The Vein Quartz + Chalcopyrite ore type was not sorted, though approximately 1 to 2 % of the particles consisted of non-mineralised, coarse-grained, white quartz. The handpicked fractions were weighed and the results are presented in Table 7.

The waste fraction constitutes 55 Wt % of the feed and consists of barren (less than 1 % visible sulfides) wallrock particles. Of this 55 % waste component approximately 60 % is greater than 10 mm (See Table 40 and Figure 15 in GRAIN SIZE ANALYSIS). Thus, approximately 33 % of the feed consists of barren waste in the size range of greater than 1 cm, and is potentially amenable to photometric ore sorting (Wills 1985). This coarse fraction could be separated using a combination of screens and/or cyclones. Furthermore, based on the difference in specific gravity of the gangue minerals (approx. 2.8) and the ore minerals (approx. 4.5), pre-concentration by gravity separation is potentially feasible. Consequently, approximately one third of the feed on the February 5th could potentially have been removed prior to grinding, with resultant decrease in grinding costs, reagent costs, and possibly talcose slime production. However, it must be stressed that the grab sample, though apparently representative of the feed during the sampling campaign, may not be representative of the 'average' feed. This is particularly true in view of the higher than average Cu assays and the abundance of the Chalcopyrite Stringer ore, which is likely to have contributed to a higher wallrock component in the rod mill feed. However, the author contends that the magnitude of this wallrock component is unlikely to fall below the 25 % level.

The hand picked fractions indicate that indeed the principal ore type is the Chalcopyrite Stringer, with lesser equivalent amounts of the Banded Pyrite + Sphalerite and the Massive Pyrite ore types, and is thereby in general agreement with the relative abundances of the ore types calculated by material balance.

TABLE 7

THE ORE TYPE ABUNDANCES IN THE MILL FEED AS DETERMINED BY HAND SORTING.

FRACTION.	WEIGHT %.	NORMALISED WEIGHT %. (Waste-free)
Waste.	55 %	----
Cpy Stringer.	22 %	49 %
Banded Py + Sphal.	10 %	23 %
Massive Py.	10 %	21 %
Mixed/Banded Cpy + Sphal.	2 %	4 %
Massive Sphal.	1 %	3 %

Note : The Sheared Chalcopyrite + Sphalerite ore type most probably contributed to the Mixed/Banded Chalcopyrite + Sphalerite and Chalcopyrite Stringer ore types. The Disseminated Chalcopyrite + Pyrite ore type most probably contributed to the Chalcopyrite Stringer and the Massive Pyrite ore types.

Chapter V

CHEMICAL ANALYSIS

The 25 samples were chemically analysed for 19 elements, and the assays, except those for Bi which were below the limit of detection (< 0.001 Wt % Bi), are presented in Table 8. The samples are ordered for presentation starting with the float feed, the Cu-circuit concentrates, the Zn-circuit feed, the Zn-circuit concentrates, the Cu-circuit tailings and the Zn-circuit tailings. This format will be consistently used throughout the report. The analyses for Ni indicate concentrations of between 0.001 and 0.002 Wt % Ni, where 0.001 Wt % Ni is the detection limit. Thus, the variance exhibited by Ni is of the same order as the analytical uncertainty, due to the proximity to the detection limits. One analysis for Sn was below the detection limit of 1 ppm Sn, whereas 6 analyses for In and 12 for Te were below the detection limits of 10 ppm and 0.2 ppm, respectively. The remaining 14 elements indicated concentrations for all samples above their respective detection limits. For the purpose of material balance calculations, half the value of the detection limit was substituted for data points that were below the detection limit (i.e. 'nd.'). This procedure provides relatively realistic first approximations to the true values, which the material balance adjustment is attempting to approach. Therefore, the adjusted assays and calculated recoveries for Sn, Ni, Te and particularly In contain larger uncertainties carried over from the raw data.

The assays were adjusted by material balance calculations in order to remove analytical errors, and model the steady-state operation of the circuit. The adjusted assays and the calculated weights of each product are presented in Table 9. The recovery of the 18 elements in each of the 17 products was calculated from these adjusted assays and are presented in Table 10. The recovery is the percentage of the total mass of a specific element in a specific product, and accurately describes the distribution of the element in the feed among the various products, whether it be concentrates or tailings. The adjusted assays and recoveries for the valuable elements and iron are plotted on circuit diagrams, and are given in Appendix A. Histograms of the recovery of each element (except Bi) in each product are presented in Figure 2, and graphically describe the distribution and behaviour of each element through the circuit. Elements with similar distributions are readily grouped by simple visual comparison of these histograms. Five groups of elements are recognised.

Group 1 comprises the four elements Mn, Ni, Co and Fe, whose behaviour is essentially inert and report principally (> 80 %) to the tailings. Group 2 comprises the five elements As, Sb, Hg, Se and S, whose behaviour is partly inert with between 55 and 80 % of each of the elements reporting to the tailings. In addition, these elements exhibit comparable recoveries (between 10 and 40 %) to the Cu- and Zn-rougher concentrates, and which tends to be

TABLE 8

THE RAW CHEMICAL ASSAYS FOR THE 25 TROUT LAKE MILL PRODUCT SAMPLES.

SAMPLE.	Au	Ag	Cu	In	Ni	Cd	Co	Mn	Pb	Fe	S	As	Hg	Te	Se	Sn	Sb	In
	ppm		Wt %										ppm					
TL.MP-1	2.195	12.05	2.19	2.7	0.001	0.006	0.012	0.090	0.037	16.1	9.7	0.06	22.	0.3	53.	8.	30.	nd.
TL.MP-8	52.691	94.33	8.45	11.1	0.001	0.021	0.028	0.041	0.122	29.3	36.2	0.28	85.	5.7	112.	46.	130.	31.
TL.MP-2	24.010	79.23	17.87	5.9	0.001	0.013	0.019	0.026	0.103	28.8	33.0	0.21	46.	7.0	158.	53.	98.	54.
TL.MP-4	15.847	65.17	17.37	4.8	0.002	0.010	0.015	0.032	0.084	27.2	29.7	0.15	40.	5.0	148.	53.	93.	45.
TL.MP-6	23.393	78.20	18.88	5.7	0.002	0.013	0.017	0.023	0.098	28.7	32.8	0.14	42.	4.0	159.	55.	107.	48.
TL.MP-15	22.707	78.55	16.77	6.1	0.001	0.014	0.018	0.027	0.108	28.2	31.7	0.18	46.	7.3	144.	66.	106.	63.
TL.MP-10	35.260	100.16	25.20	4.4	0.002	0.010	0.015	0.010	0.081	30.3	35.5	0.08	44.	4.8	173.	56.	89.	55.
TL.MP-12	15.778	81.98	28.41	4.2	0.001	0.010	0.012	0.006	0.068	29.6	35.0	0.04	38.	5.7	179.	67.	89.	77.
TL.MP-21	2.470	6.86	0.20	2.4	0.001	0.005	0.010	0.094	0.044	15.2	8.1	0.87	17.	nd.	43.	8.	31.	nd.
TL.MP-14	8.332	50.42	1.64	38.9	0.001	0.078	0.019	0.043	0.360	16.6	31.0	0.30	79.	nd.	89.	43.	182.	81.
TL.MP-16	6.379	46.99	1.43	32.0	0.002	0.070	0.016	0.051	0.300	16.9	28.8	0.30	73.	nd.	78.	43.	154.	67.
TL.MP-18	8.232	46.65	1.51	28.1	0.001	0.060	0.019	0.055	0.300	19.1	29.1	0.34	70.	nd.	93.	38.	165.	50.
TL.MP-20	7.889	52.17	1.65	37.2	0.001	0.083	0.015	0.040	0.360	16.2	31.3	0.34	82.	nd.	92.	35.	185.	50.
TL.MP-22	8.026	52.82	1.66	33.9	0.001	0.073	0.018	0.045	0.340	17.5	31.1	0.30	85.	nd.	85.	33.	185.	56.
TL.MP-24	11.113	59.00	2.01	32.8	0.001	0.071	0.019	0.048	0.440	19.0	32.9	0.37	80.	nd.	87.	49.	168.	98.
TL.MP-26	6.929	61.05	1.56	46.2	0.001	0.102	0.012	0.029	0.440	12.6	32.7	0.27	100.	nd.	80.	40.	195.	53.
TL.MP-3	1.989	8.92	0.35	2.7	0.001	0.006	0.009	0.095	0.038	15.3	8.5	0.42	17.	nd.	38.	3.	32.	nd.
TL.MP-7	0.686	6.52	0.20	3.4	0.001	0.008	0.010	0.094	0.034	16.8	7.5	0.17	16.	nd.	46.	nd.	25.	nd.
TL.MP-9	0.755	6.52	0.18	2.7	0.001	0.005	0.011	0.096	0.036	14.9	7.6	0.07	18.	nd.	43.	4.	39.	nd.
TL.MP-11	19.071	56.94	4.10	7.2	0.001	0.016	0.023	0.055	0.127	26.4	27.6	0.27	48.	3.3	94.	45.	97.	40.
TL.MP-25	21.746	66.54	6.60	7.5	0.001	0.017	0.022	0.053	0.128	26.5	28.1	0.30	50.	4.5	106.	38.	96.	28.
TL.MP-13	25.245	102.56	12.03	9.7	0.001	0.021	0.023	0.026	0.140	28.8	34.3	0.59	54.	8.7	132.	55.	126.	62.
TL.MP-23	3.293	30.18	0.89	2.4	0.001	0.005	0.024	0.075	0.310	27.5	23.2	0.41	38.	6.0	94.	29.	89.	13.
TL.MP-19	11.388	43.90	2.13	5.6	0.001	0.013	0.026	0.078	0.830	33.1	33.8	0.45	51.	0.3	113.	60.	113.	52.
TL.MP-17	0.823	4.46	0.12	14.9	0.001	0.001	0.010	0.097	0.017	14.9	6.2	0.06	15.	nd.	33.	nd.	22.	nd.

'nd.' denotes not detected.

Bi analyses all below the detection limit of 0.001 Wt % Bi, and thus are not presented.

TABLE 9

THE ADJUSTED ASSAYS FOR THE 17 TROUT LAKE MILL PRODUCTS.

SAMPLE.	PRODUCT WEIGHT.	Au	Ag	Cu	Zn	Ni	Cd	Co	Mn	Pb	Fe	S	As	Hg	Te	Se	Sn	Sb	In
		ppm		Wt %										ppm					
1. Feed.	100.000	2.23	12.35	2.16	2.57	0.001	0.006	0.010	0.089	0.038	15.96	9.58	0.075	19.	0.4	49.	7.	34.	8.
2. Cu-Scav Co.	1.925	52.41	94.94	8.36	11.12	0.001	0.021	0.028	0.041	0.122	29.31	34.30	0.283	55.	3.0	112.	44.	130.	31.
3 Cu-Roug Co.	6.246	19.59	73.78	17.40	5.41	0.001	0.012	0.017	0.079	0.095	28.13	31.54	0.156	43.	4.6	147.	55.	94.	45.
4. Cu-Cl.1 Fd.	13.583	20.00	75.67	17.13	5.68	0.001	0.013	0.017	0.028	0.098	28.17	31.71	0.179	44.	4.8	146.	55.	97.	46.
5. Cu-Cl.1 Co.	7.909	19.48	86.92	26.29	4.60	0.001	0.011	0.014	0.009	0.077	29.63	34.80	0.090	41.	5.0	179.	64.	93.	53.
6. Cu.Cl.2 Co.	7.064	18.69	84.86	28.00	3.99	0.001	0.009	0.013	0.006	0.070	29.72	34.85	0.038	40.	4.8	184.	65.	89.	51.
7. Zn-Roug Fd.	94.756	1.03	7.31	0.21	2.46	0.001	0.006	0.010	0.095	0.040	15.16	7.95	0.084	18.	0.1	40.	3.	31.	5.
8. Zn-Roug Co.	6.305	6.38	51.24	1.45	34.34	0.001	0.069	0.016	0.044	0.360	16.99	30.11	0.322	79.	0.1	85.	35.	167.	42.
9. Zn-Cl.1 Co.	6.817	9.12	54.71	1.85	33.04	0.001	0.067	0.018	0.047	0.807	19.22	33.17	0.346	80.	0.1	91.	45.	169.	54.
10. Zn-Cl.2 Co.	4.485	7.58	59.92	1.67	47.29	0.001	0.096	0.013	0.030	0.381	12.64	32.90	0.285	96.	0.1	81.	36.	198.	53.
11. Cu-Roug Tl.	94.861	2.02	8.64	0.36	2.64	0.001	0.006	0.010	0.094	0.037	15.20	8.20	0.082	19.	0.1	41.	3.	32.	5.
12. Cu-Scav Tl.	92.936	1.00	6.86	0.19	2.46	0.001	0.006	0.010	0.095	0.035	14.91	7.66	0.078	18.	0.1	39.	3.	30.	5.
13. Cu-Cl.1 Tl.	5.674	20.60	59.99	4.36	7.18	0.001	0.015	0.022	0.055	0.126	26.14	27.41	0.303	48.	4.6	101.	42.	102.	37.
14. Cu-Cl.2 Tl.	0.845	26.31	104.34	11.99	9.66	0.001	0.021	0.023	0.027	0.142	28.86	34.37	0.530	55.	8.7	132.	55.	136.	63.
15. Zn-Cl.1 Tl.	1.820	3.43	29.91	0.90	2.40	0.001	0.005	0.025	0.077	0.307	27.70	23.25	0.413	38.	0.2	95.	31.	90.	13.
16. Zn-Cl.2 Tl.	2.331	12.09	44.73	2.19	5.60	0.001	0.013	0.027	0.079	0.749	31.88	33.70	0.462	51.	0.2	111.	62.	113.	56.
17. Zn-Roug Tl.	88.451	0.65	4.15	0.12	0.19	0.001	0.001	0.010	0.098	0.018	18.03	6.38	0.067	14.	0.1	37.	1.	21.	2.

TABLE 10

THE ELEMENT RECOVERIES IN THE 17 TROUT LAKE MILL PRODUCTS.

SAMPLE.	Au	Ag	Cu	Zn	Ni	Cd	Co	Mn	Pb	Fe	S	As	Hg	Te	Se	Sn	Sb	In
1. Feed.	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
2. Cu-Scav Co.	45.29	14.79	7.47	8.32	1.90	6.99	5.31	0.89	6.26	3.54	6.90	7.29	5.47	14.47	4.36	12.08	7.38	7.35
3. Cu-Roug Co.	111.99	76.02	103.21	26.80	14.94	26.19	21.20	4.00	32.04	22.25	41.95	26.57	28.62	146.84	38.06	99.83	35.22	71.39
4. Cu-Cl.1 Fd.	121.86	83.15	107.91	29.98	15.78	29.25	23.10	4.26	35.24	23.98	44.98	32.56	31.01	165.49	40.32	106.43	38.75	78.03
5. Cu-Cl.1 Co.	69.19	55.62	96.44	14.14	9.71	16.32	10.87	0.77	16.29	14.69	28.74	9.57	16.92	99.23	28.69	72.07	21.77	51.70
6. Cu.Cl.2 Co.	59.22	48.49	91.74	10.96	8.87	11.26	8.90	0.51	13.09	13.16	25.71	3.57	14.52	80.58	26.44	65.47	18.44	45.07
7. Zn-Roug Fd.	43.57	55.91	9.02	90.73	92.92	90.31	95.53	101.07	101.76	90.00	78.71	106.47	89.05	20.25	77.08	42.54	86.37	57.91
8. Zn-Roug Co.	18.03	26.14	4.23	84.17	6.21	75.52	10.03	3.11	60.36	6.71	19.83	27.15	25.84	1.70	10.85	31.22	30.97	32.57
9. Zn-Cl.1 Co.	27.91	30.18	5.84	87.54	6.71	79.20	12.24	3.60	91.97	8.21	23.61	31.53	28.44	2.17	12.58	43.86	33.92	45.85
10. Zn-Cl.2 Co.	15.25	21.74	3.47	82.47	4.61	73.96	5.59	1.52	65.52	3.55	15.41	17.10	22.27	0.88	7.34	23.20	26.17	29.59
11. Cu-Roug Tl.	86.05	66.30	15.73	97.36	93.03	95.73	96.40	100.38	93.17	90.38	81.19	103.72	90.95	33.90	77.93	46.60	88.94	62.28
12. Cu-Scav Tl.	40.78	51.51	8.26	89.04	91.13	88.74	91.10	99.49	86.91	86.84	76.29	96.43	85.48	19.43	73.56	34.53	81.56	54.93
13. Cu-Cl.1 Tl.	52.67	27.53	11.47	15.84	6.07	14.93	12.23	3.49	18.95	9.30	16.24	22.99	14.10	66.26	11.62	34.36	16.98	26.32
14. Cu-Cl.2 Tl.	9.98	7.13	4.70	3.17	0.84	3.06	1.96	0.26	3.20	1.53	3.03	6.00	2.40	18.65	2.26	6.60	3.33	6.64
15. Zn-Cl.1 Tl.	2.79	4.40	0.76	1.70	1.79	1.57	4.43	1.59	14.85	3.16	4.42	10.05	3.58	0.82	3.52	8.02	4.80	2.98
16. Zn-Cl.2 Tl.	12.66	8.43	2.37	5.07	2.29	5.25	6.64	2.07	46.46	4.66	8.20	14.42	6.17	1.30	5.24	20.66	7.75	16.25
17. In-Roug Tl.	25.54	29.77	4.80	6.57	86.72	14.79	85.50	97.97	41.40	83.29	58.88	79.33	63.21	18.55	66.23	11.33	55.39	25.34

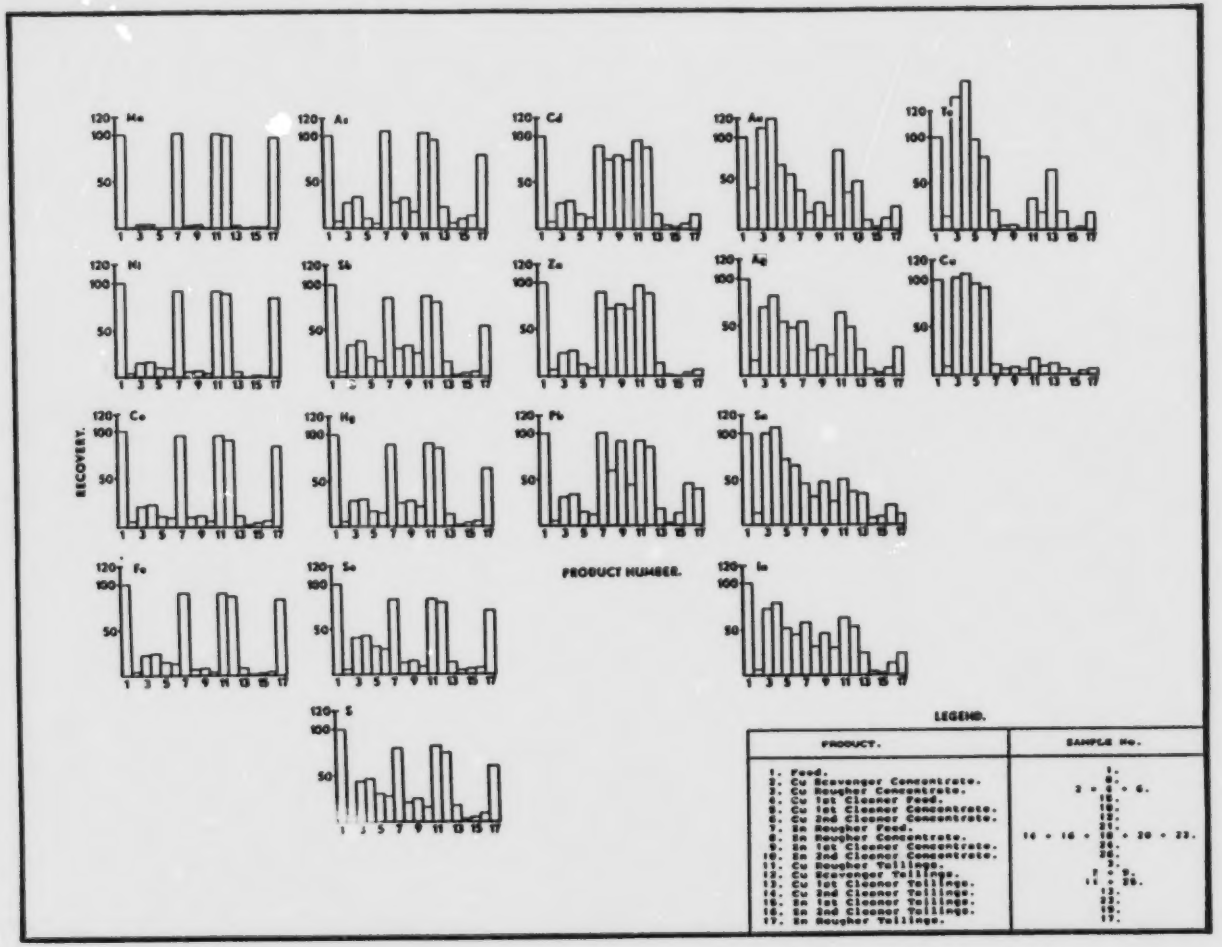

FIGURE 2.
HISTOGRAMS GIVING THE DISTRIBUTION OF THE RECOVERIES CALCULATED BY MATBAL FOR EACH ELEMENT IN THE 17 TROUT LAKE MILL PRODUCTS.

partly decreased in both the cleaning circuits. Group 3 comprises the three elements Cd, Zn and Pb, which exhibit identical behaviour in the Cu-circuit with minor recoveries in the Cu-rougher concentrates, and which is considerably reduced (by approximately 30 %) through the Cu-cleaning circuit. These elements preferentially concentrate in the Zn-circuit concentrates, though in contrast, Pb exhibits relatively high rejection in the Zn-rougher and the Zn-2nd cleaner, and thus exhibits moderately high loss to the tailings. Group 4 comprises the four elements Au, Ag, Sn and In, which display high recovery in the Cu-rougher concentrates, and which are partly reduced through the Cu-cleaning circuit, principally in the Cu-1st cleaner. These elements also exhibit moderate recovery in the Zn-circuit concentrates and the tailings. Group 5 comprises the two elements Cu and Te, which are principally recovered in the Cu-circuit concentrates, with slight reduction through the Cu-cleaning circuit. The distribution of the Te recoveries is distorted by Te concentrations in most of the Cu-circuit tailings and all the Zn-circuit products being below the detection limit, and which were thus set at half the detction limit. These values when adjusted imply anomalously high circulating recoveries of Te in the Cu-circuit. However, as Te was only detected in Cu-circuit products, it is reasonable to assume that the distribution of Te recoveries is essentially valid.

The circuit balance of the feed and the three main products for this study incorporates in tabular form, much of the information in these histograms, and is presented in Table 11. The most striking feature of Table 11 is that the Cu-concentrate assays 28.00 % Cu, and the Zn-concentrate assays 47.29 % Zn, constituting 81 % and 79 % respectively, of the maximum attainable grade based on the known average composition of the Trout Lake chalcopyrite and sphalerite (Healy & Petruk 1988). The 28.00 % Cu assay in the Cu-concentrate is a considerable improvement over the 22.85 % Cu observed in 1984, though the recovery of 91.74 % is comparable to the 92.77 % observed in 1984 (Healy & Petruk 1988). The Zn recovery in the Zn-concentrate of 82.47 % marks an improvement over the 77.25 % recorded in 1984. The Cu-concentrate assays 3.99 % Zn, representing 10.96 % recovery of Zn, an improvement over the 13.78 % recorded in 1984 (Healy & Petruk 1988). However, it is this Zn recovery/loss to the Cu-concentrate, whose resolution is the principal motivation of this study. Similarly, resolution of the high loss of Au and Ag to the tailings, 25.54 % and 29.97 % respectively, is another primary objective of this study. Circuit diagrams portraying the assays and recoveries of the valuable metals and of Fe in each product are given in Figures 1 to 6 of appendix A.

The Group 1 elements Mn, Ni, Co and Fe all exhibit high recoveries in the tailings and reflects the behaviour of the inert minerals pyrite, pyrrhotite, gangue, etc., where pyrite and pyrrhotite are the principal hosts of the Fe, Co and Ni, and the gangue is the host of the Mn (Healy & Petruk 1988).

The Group 2 elements As, Sb, Hg, Se and S report principally to the tailings, but exhibit moderate recoveries in the Cu- or Zn-concentrates. Sulfur being essentially the sole anion of the sulfides will report to all products in proportion to their sulfide content and product weight. Selenium reports largely to the tailings, with 26.44 % reporting to the Cu-concentrate, reflecting the behaviour of the principal probable hosts chalcopyrite, pyrite and pyrrhotite. The Se was expected to report principally to the Cu-concentrate, as chalcopyrite was interpreted to be the principal Se-carrier,

followed by pyrite and pyrrhotite (Healy & Petruk 1988). It is interpreted on the basis of the above recoveries that chalcopyrite, pyrite, pyrrhotite and clausthalite are the principal hosts. Arsenic reports principally to the tailings with 17.10 % recovery in the Zn-concentrate. This agrees favourably with the prediction of Healy & Petruk (1988) based on the inert arsenopyrite being the sole As-carrier of significance, yet the occurrence of fine-grained arsenopyrite disseminations in sphalerite contributing sphalerite-arsenopyrite composite particles that will partly report to the Zn-concentrate.

Hg reports principally to the tailings (63.2 % recovery) with 14.52 % in the Cu-concentrate and 22.27 % in the Zn-concentrate. The accuracy of the Hg analyses had been questioned by Healy & Petruk (1988), who suggested that the ease with which the Hg assays for the mill products balanced (minimum adjustment to raw assays by MATBAL) would indicate the accuracy of the analyses. The Hg assays balanced readily with only slight adjustments (See Tables 8 and 9). Thus, the Hg assays are interpreted to be accurate and indicate that sphalerite the sole Zn-carrier of significance, hosts a maximum of 27 % of the Hg in the ores (i.e. the Zn-concentrate contains 82 % recovery of the Zn, but only 22 % recovery of the Hg). The similarity of the distributions of the Hg and Sb recoveries (See Table 10 and Figure 2) suggests that the Sb-sulfosalts host the bulk of the Hg, while the mercurian electrum is known to carry approximately 2 to 3 % of the Hg in the ore (Healy & Petruk 1988). However, as the Hg content of the feed is approximately half that of the Sb, and as Hg was not observed as a major, minor or trace element in any of the Sb-sulfosalts, the principal host of the Hg remains unresolved. Sb also reports principally to the tailings with lesser recoveries in the Zn- and Cu-concentrates. Eight Sb-bearing minerals were observed in the ore, of which freibergite, gudmundite and pyrargyrite are the dominant minerals, two of which are Ag-bearing (Healy & Petruk 1988). Because of the diversity of Sb-minerals it would be premature (in this chapter) to attempt to describe the distribution of Sb in the circuit in terms of the behaviour of these minerals. However, it may be expected that gudmundite and possibly pyrargyrite will report principally to the tailings, whereas freibergite will report to the Cu-concentrate, the tailings and the Zn-concentrate, in that order of abundance.

The three Group 3 elements Zn, Cd and Pb report principally to the Zn-concentrate, though Pb exhibits the highest losses to the tailings i.e. 41.40 % Pb recovery in the tailings. Zn and Cd are carried in sphalerite, which reports to the Zn-concentrate after activation with $CuSO_4$, though approximately 11 % of the sphalerite reports to the Cu-concentrate prior to that stage. The higher apparent Cd losses to the tailings may reflect greater analytical uncertainty in the Cd analyses or that a second subordinate Cd-bearing mineral occurs in the ores. The latter is highly improbable in view of the known universal geochemical coherence of Zn and Cd, and the high Zn-Cd correlation coefficient of + 0.962 observed for the assays of the Trout Lake ores (Healy & Petruk 1988). The Pb is principally hosted in galena, which therefore tends to report to the Zn-concentrate and the tailings, in response to it's flotation characteristics and that of the various host minerals with which it is expected to form unliberated composite grains. Clearly the galena does not readily report to the Cu-concentrate as observed in many other processing plants (Petruk 1985). However, this behaviour may not reflect the flotation characteristics of galena, which was observed to be fine-grained in the broken ore (Healy & Petruk 1988) and which is largely unliberated (See MINERAL LIBERATIONS).

The four Group 4 elements Au, Ag, Sn and In report principally to the Cu-concentrate, but also exhibit moderate recoveries in the tailings and Zn-concentrates. Cassiterite the only Sn-bearing mineral identified in the ores (Healy & Petruk 1988) is expected to report largely to the tailings based on it's known behaviour during floatation (Petruk 1985). In addition, the cassiterite was interpreted to be largely associated with the sphalerite and to a lesser extent with the chalcopyrite, and thus if unliberated was expected to partly report to the Zn-concentrate in the form of sphalerite-cassiterite composite grains. Cassiterite associated with chalcopyrite and possibly stannite or Sn in solid solution in chalcopyrite were expected to partly report to the Cu-concentrate. The 65.47 % recovery of Sn in the Cu-concentrate suggests that the Sn occurring in solid solution in chalcopyrite, or in cassiterite or possibly in stannite associated with chalcopyrite predominates over the cassiterite associated with sphalerite.

Mercurian electrum, the sole Au-bearing mineral identified in the Trout Lake ores is fine-grained and partly hosted in pyrite (Healy & Petruk 1988), and is thus expected to be partly unliberated. It should therefore report to the Cu-concentrate as free electrum or electrum locked with chalcopyrite, or to the tailings as pyrite-electrum composite grains, or possibly as Au, either in solid solution, or as submicroscopic inclusions in pyrite. The loss of 15.25 % Au to the Zn-concentrate probably reflects the common association of electrum and sphalerite filling fractures in pyrite, which if liberated from the pyrite may remain as sphalerite-electrum composite grains, of which many would be expected to report to the Zn-concentrate. In addition, composite grains of sphalerite-pyrite may contain fine-grained inclusions of electrum in the pyrite, or Au in solid solution in the pyrite.

Ten Ag-bearing minerals were identified, while sphalerite and chalcopyrite are also probably Ag-carriers. The principal Ag-carriers are interpreted to be freibergite, sphalerite, pyrargyrite, galena, chalcopyrite and electrum (Healy & Petruk 1988). Ag occurring in liberated freibergite, pyrargyrite chalcopyrite and electrum is expected to report to the Cu-concentrate, Ag in liberated sphalerite will report to the Zn-concentrate, while the behaviour of liberated galena and hence Ag in that galena is unresolved at this point. The behaviour of all these Ag-carriers which remain unliberated is defined by the flotation characteristics of the host mineral(s). The multiplicity of Ag-bearing minerals, the fine-grained nature of many of them, and the potential number of locked species containing the Ag-carriers contributes to the lack of specificity in the recovery of Ag, though there is a greater tendancy for recovery in the Cu-concentrate. The similarity in the distributions of the Ag and In recoveries is striking, and suggests that the bulk of the Ag occurs in solid solution in chalcopyrite and sphalerite due to the coupled substitution $Me^{2+} = In^{3+} + Ag^{1+}$, which has been observed in synthetic sphalerites (McIntyre et al. 1984).

The two Group 5 elements Cu and Te are principally recovered in the Cu-concentrate (91.74 and 80.58 % recovery, respectively) reflects the excellent recovery of chalcopyrite, the sole Cu-carrier of significance, and the interpreted principal host of the Te in the form of Te in solid solution in chalcopyrite. In addition, the four Te-bearing minerals hessite, rucklidgeite, pilsenite and volynskite are fine-grained and occur associated with, and generally occur as inclusions in, chalcopyrite, and are therefore expected to

report to the Cu-concentrate if unliberated. The flotation characteristics of these Te-minerals is unknown, and thus the behaviour of free grains of these minerals is unpredictable at present.

TABLE 11

THE TROUT LAKE CIRCUIT BALANCE ON FEBUARY 5th 1986.

PRODUCT.	PRODUCT WEIGHT.	ASSAYS.																
		Au	Ag	Cu	Zn	Ni	Cd	Co	Mn	Pb	Fe	S	As	Hg	Te	Se	Sn	Sb	In
		ppm		Wt %										ppm					
Feed.	100.000	2.230	12.348	2.16	2.07	0.001	0.006	0.010	0.089	0.038	15.96	9.58	0.075	19.3	0.40	49.2	7.00	33.9	8.06
Cu-Conc.	7.064	18.694	84.858	28.00	3.99	0.001	0.009	0.013	0.006	0.070	29.72	34.85	0.038	39.6	4.51	184.3	64.88	88.6	51.44
Zn-Conc.	4.485	7.580	59.922	1.67	47.29	0.001	0.096	0.013	0.030	0.381	12.64	32.90	0.285	95.6	0.08	80.6	36.21	198.0	53.19
Tails.	88.451	0.652	4.150	0.12	0.19	0.001	0.001	0.010	0.098	0.018	15.03	6.38	0.067	13.8	0.08	36.9	0.89	21.3	2.31
				RECOVERY (%)															
Cu-Conc.		59.22	48.49	91.74	10.96	8.87	11.26	8.90	0.51	13.09	13.16	25.71	3.57	14.52	80.58	26.44	68.47	18.44	45.07
Zn-Conc.		15.25	21.74	3.47	82.47	4.41	73.96	5.59	1.52	45.59	3.55	15.41	17.10	22.27	0.88	7.34	23.20	26.17	29.59
Tails.		25.54	29.77	4.80	6.57	86.72	14.79	85.56	97.97	41.40	83.29	58.88	79.33	63.21	18.55	66.23	11.33	55.39	25.34

CONCENTRATION FACTORS

As the assays and recovery values represent non-normally distributed variables this dataset is not amenable to standard statistical techniques. A technique for comparing assay values in different products which involves calculation of 'Concentration Factors' (Petruk & Schnarr 1981) was used. The technique involves (a) dividing the assay value for each element by the assay value for every other element in the feed, (b) performing the same for each product, and (c) then dividing the set of values generated for each product by the equivalent set generated for the feed (Petruk & Schnarr 1981). The resulting value is the concentration factor, and it compares the concentration of each element to that of every other element in that sample as well as to that of the feed. This factor is therefore the relative concentration of an element compared to that of the feed. A concentration factor of 1 indicates no relative concentration, while a concentration factor of greater than one indicates relative concentration, and a concentration factor of less than one indicates relative depletion.

This technique requires a computer program where large datasets (i.e. greater than 8 or 10 elements) are involved. The author wrote a FORTRAN 77 program called CONFAC to perform this task, as such a program was not then available. Concentration factors were calculated for 16 the elements; Bi, Ni and S were excluded. The concentration factors, which are given in Table 12 were plotted to produce discontinuous curves or each element from the Cu-circuit concentrates, to the Cu-circuit tailings, to the Zn-circuit concentrates, to the Zn-circuit tailings.

The Cu curves show the behaviour of elements that are concentrated in the Cu-concentrate; Zn curves the behaviour of elements that are concentrated in the Zn-concentrate; and Mn curves the behaviour of those rejected to the tailings (i.e. silicate and carbonate gangue). Comparison of curves for other elements to the Cu, Zn and Mn curves indicate the tendency of other elements to concentrate in the Cu-concentrate, Zn-concentrate and the tailings. Because sphalerite is partly recovered in the Cu-concentrate, the Zn curves are not entirely representative of an element that reports to the Zn-concentrate.

The Zn curves show an unusual behaviour in the Cu-circuit. Both Zn and Mn should be rejected from the Cu-circuit, and hence their curves in the Cu-circuit should be identical. However, the Zn shows moderately low relative concentration, whereas Mn shows the strongest relative depletion of all 16 elements. Furthermore, in comparison to Mn, Cu and Fe, there is a marked concentration of Zn in the Cu-scavenger concentrate, suggesting that in the Cu-scavenger cells, sphalerite is preferentially floated over chalcopyrite, gangue and pyrite. It is also observed that the moderately low relative concentration of Zn through the Cu-circuit (except in the Cu-scavenger concentrate) is the same as that for Pb. In the Cu-concentrate, the recoveries of Zn, which is essentially sphalerite, and of Pb which is essentially galena, are 11.0 % and 13.1 %, respectively. The Zn and Cd curves are virtually identical, and thus confirms that sphalerite is the sole host of Cd.

The Fe and Co curves are similar to those of Mn, with the exception of in the Cu-circuit concentrates. Both Fe and Co exhibit moderate relative concentration in these concentrates, and probably reflects the Fe component in

chalcopyrite, and the recovery of Fe and Co in pyrite which is partly recovered in the Cu-concentrate. However, as Fe is principally hosted in pyrite and gangue; Co in pyrite and pyrrhotite; and Mn in gangue, where these are expected to be rejected to the tailings, these elements show the strongest relative concentration of the 16 elements in the tailings.

Gold exhibits a relative concentration in all products, except the Cu-scavenger tailings (i.e. CF = 0.95), with the highest concentration in the Cu-circuit concentrates. The Au concentration factor curves do not correspond to those of any other element, though they partly resemble that of Cu. However, Au shows the strongest relative concentration in the Cu-concentrate (i.e. CF = 11.4), and is notably enriched in the Cu-scavenger concentrate and the Cu 1st and 2nd cleaner tailings. The Au curves probably reflect the recovery in the Cu-circuit concentrates of free electrum and electrum locked with chalcopyrite; the recovery in the Zn-circuit concentrates of electrum locked in composite particles with sphalerite and pyrite; and recovery in the tailings of fine-grained inclusions entrapped in pyrite, and possibly Au in solid solution in pyrite and/or arsenopyrite. The unusually high relative concentration of Au in the Cu-scavenger concentrate and the Cu cleaner tailings indicates the recovery of high proportion of Au-bearing particles of an unknown character. Possibly these correspond to coarse-grained free type 2 electrum which is recovered by the 'hard pulling' Cu-scavenger, but rejected by the Cu-cleaners, resulting in high circulating loads for Au in the Cu-circuit. As mercurian electrum has a mean specific gravity of approximately 15, the probable maximum grain size for optimum flotation is possibly only 30 to 50 um, as opposed to 100 um for chalcopyrite.

The Ag curve for the Cu-circuit concentrates resembles that of Cu, and for the Zn-circuit concentrates resembles that of Zn, though the magnitude of relative concentration of Ag is markedly less than that of Cu and Zn in the respective circuit concentrates. This suggests that Ag is hosted in chalcopyrite (and freibergite, which has similar flotation characteristics to chalcopyrite) and in sphalerite. Even more striking is the similarity in the Ag and In curves in both the Cu- and Zn-circuit concentrates. The strength of this relationship suggests that both In and Ag occur largely in solid solution in chalcopyrite and sphalerite (See Healy & Petruk 1988). It is noteworthy that Ag is enriched in the Cu-scavenger concentrate, probably reflecting Ag in electrum, which is preferentially recovered in this product. Ag also exhibits relative concentration in all 17 products, probably reflecting the multiplicity of Ag-bearing minerals, and the influence of host minerals on the unliberated fine-grained Ag-bearing mineral inclusions.

The concentration factor curves for Pb show an enrichment in the Zn-circuit concentrates and tailings, particularly the Zn-2nd cleaner tailings, but do not correspond to the Zn or Mn (or Fe) curves, though Pb approximates to that of Zn in many products. A logical interpretation is that the Pb is due to galena locked with sphalerite and pyrite. The combination of locked galena grains in particles containing these minerals obscures the relationship to Zn and Fe. In this regard, it is noteworthy that Zn, Pb and Fe have virtually identical behaviour in the Cu-circuit products.

The concentration factor curves for Te, Se, Sn and In exhibit a general similarity, except for the high relative concentration of Te in the Cu-cleaner

tailings, but none corresponds to those of Cu, Zn or Mn. Te exhibits high relative concentration in the Cu-circuit products, and relative depletion in the Zn-circuit products, reflecting the high recovery of Te in solid solution in chalcopyrite and in various tellurides (See Table 13). The high circulating loads of Te in the Cu-cleaning circuit probably reflects the latter, as this trend is not observed for Cu and hence chalcopyrite. Sn and In exhibit almost identical curves, except for the higher relative concentration of Sn in the Cu-circuit concentrates and In in the Zn-circuit concentrates. The In curves are interpreted to reflect In in solid solution in both chalcopyrite and sphalerite, whereas the Sn curves reflect principally Sn in solid solution in chalcopyrite, and the recovery of Sn in cassiterite locked with chalcopyrite to the Cu-concentrate, and locked with sphalerite to the Zn-concentrate. The Se curves exhibit comparatively moderate to low relative concentration in almost all products, with the highest relative concentration in the Cu-circuit concentrates and the Zn-rougher tailings. The Se curves indicate that chalcopyrite is the principal host of Se, and that Se in pyrite and pyrrhotite principally reports to the Zn-rougher tailings, whereas Se in galena-clausthalite solid solution probably reports to the Zn-rougher tailings and the Zn-concentrate in equivalent amounts.

The concentration factor curves for Hg, As and Sb are generally similar, and exhibit low to moderate relative concentration or relative depletion in all products. The curves resemble that of Zn in the Cu-circuit concentrates and tailings, with the exception of As, which is relative depleted in the Cu-circuit concentrates. The latter reflects the rejection of arsenopyrite and possibly arsenian pyrite (See ARSENIC MINERAL-METAL ACCOUNTING) from the Cu-circuit. In the Zn-circuit the As, Sb and Hg curves do not resemble any other element, and show moderate relative concentration in the concentrate and tailings. Of these 3 elements, As exhibits the lowest relative concentration in the Zn-circuit concentrates and the highest relative concentration in the Zn-circuit tailings, reflecting the rejection of arsenopyrite and possibly arsenian pyrite to the tailings. As reporting to the Zn-circuit concentrate probably reflects sphalerite-arsenopyrite composite particles, which on the basis of textural association Healy & Petruk (1988) predicted would be relatively abundant in the ground products. Hg and Sb exhibit very similar curves in the Zn-circuit, with slightly higher relative concentration of Sb in the Zn-concentrates, and Hg in the tailings. Because of the multiplicity of fine-grained and probably largely unliberated Sb-bearing minerals, the behaviour of Sb is essentially non-specific. The distinct lack of conformity of the Hg and Zn curves in the Zn-circuit, confirms that sphalerite is not the principal host of Hg. The principal host of Hg is unknown, but is most probably pyrite.

The highest Cu concentration factor is recorded in the Cu- concentrate and has a value of 17.6, whereas that of Zn is recorded in the Zn-concentrate, and has a value of 14.9. This reflects the enrichment ratios of 13.0 for Cu and 18.4 for Zn, being inversely related to the recovery and economic value of these two valuable metals.

TABLE 12

CONCENTRATION FACTORS FOR 16 ELEMENTS IN THE 17 MILL PRODUCTS.

PRODUCT.	Au	Ag	Cu	Zn	Cd	Co	Mn	Pb	Fe	As	Hg	Te	Se	Sn	Sb	In
1. Feed	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
2. Cu-Scav Co.	9.09	2.97	1.50	1.72	1.35	1.08	0.18	1.24	0.71	1.46	1.12	2.90	0.88	2.43	1.48	1.50
3. Cu-Roug Co.	3.33	2.26	3.05	0.82	0.76	0.64	0.34	0.95	0.67	0.79	0.86	4.36	1.14	2.98	1.05	2.13
4. Cu-Cl.1 Fd.	4.45	3.04	3.94	1.13	1.08	0.84	0.16	1.28	0.88	1.19	1.15	5.96	1.48	3.90	1.42	2.86
5. Cu-Cl.1 Co.	8.37	6.75	11.67	1.76	1.76	1.34	0.10	1.94	1.78	1.15	2.07	11.98	3.50	8.77	2.62	6.35
6. Cu-Cl.2 Co.	11.40	9.35	17.64	2.17	2.04	1.77	0.09	2.51	2.53	0.69	2.86	15.31	5.11	12.63	3.56	8.67
7. Zn-Roug Fd.	0.93	1.19	0.20	1.99	2.02	2.02	2.15	2.12	1.92	2.26	1.91	0.50	1.65	0.87	1.84	1.26
8. Zn-Roug Co.	2.07	3.00	0.49	9.94	8.32	1.16	0.36	6.85	0.77	3.11	3.01	0.18	1.26	3.62	3.55	3.80
9. Zn-Cl.1 Co.	2.72	2.95	0.57	8.80	7.44	1.20	0.35	8.89	0.80	3.07	2.80	0.17	1.24	4.28	3.31	4.50
10. Zn-Cl.2 Co.	2.67	3.82	0.61	14.89	12.59	1.02	0.27	7.89	0.62	2.99	3.98	0.20	1.30	4.05	4.58	5.21
11. Cu-Roug Tl.	1.51	1.16	0.28	1.76	1.66	1.66	1.76	1.62	1.58	1.82	1.66	0.42	1.39	0.71	1.57	1.04
12. Cu-Scav Tl.	0.95	1.18	0.19	2.08	2.12	2.12	2.26	1.95	1.98	2.20	2.00	0.53	1.68	0.91	1.87	1.32
13. Cu-Cl.1 Tl.	3.74	1.97	0.82	1.16	1.01	0.89	0.25	1.34	0.66	1.63	1.02	4.65	0.83	2.43	1.21	1.87
14. Cu-Cl.2 Tl.	5.09	3.49	2.39	1.67	1.51	0.99	0.13	1.61	0.78	3.05	1.25	9.38	1.16	3.39	1.70	3.39
15. Zn-Cl.1 Tl.	1.19	1.87	0.32	0.74	0.64	1.93	0.67	6.25	1.34	4.26	1.55	0.39	1.50	3.43	2.05	1.26
16. Zn-Cl.2 Tl.	2.69	1.80	0.50	1.11	1.07	1.34	0.44	9.78	0.99	3.06	1.33	0.25	1.12	4.39	1.65	3.47
17. Zn-Roug Tl.	1.26	1.45	0.24	0.33	0.72	4.32	4.75	2.04	4.07	3.86	3.18	1.08	3.26	0.62	2.67	1.08

Note : Both Bi, Ni and S were omitted from the dataset analysed by the program CONFAC.

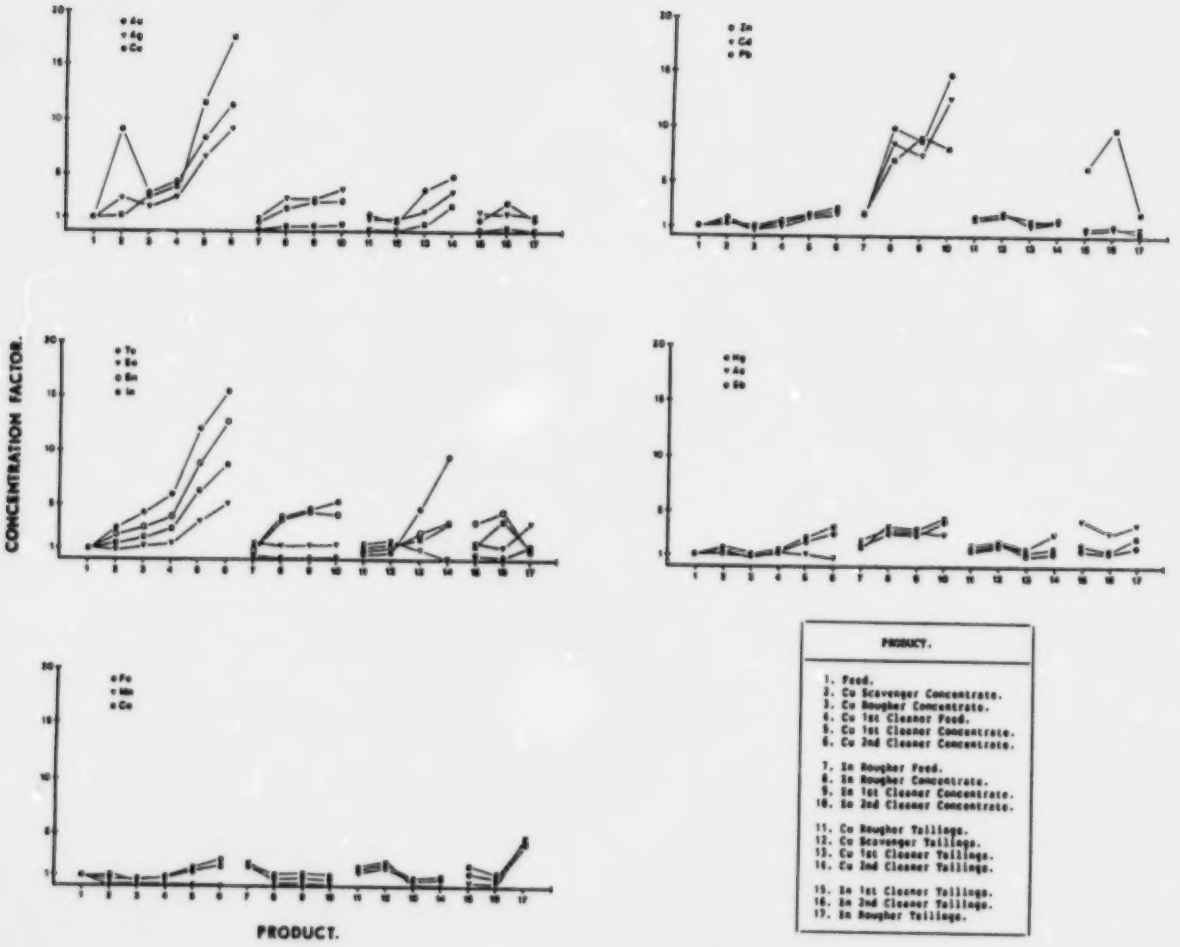
CONCENTRATION FACTOR.
PRODUCT.
PRODUCT.
1. Feed.
2. Cu Scavenger Concentrate.
3. Cu Rougher Concentrate.
4. Cu 1st Cleaner Feed.
5. Cu 1st Cleaner Concentrate.
6. Cu 2nd Cleaner Concentrate.
7. In Rougher Feed.
8. In Rougher Concentrate.
9. In 1st Cleaner Concentrate.
10. In 2nd Cleaner Concentrate.
11. Cu Rougher Tailings.
12. Cu Scavenger Tailings.
13. Cu 1st Cleaner Tailings.
14. Cu 2nd Cleaner Tailings.
15. In 1st Cleaner Tailings.
16. In 2nd Cleaner Tailings.
17. In Rougher Tailings.

FIGURE 3.
CONCENTRATION FACTOR CURVES FOR THE ELEMENTS IN THE 17 MILL PRODUCTS.

Chapter VI

MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES

This chapter describes the evaluation of the abundances in the feed and various products, of some of those minerals that were observed in the Trout Lake ores (Healy & Petruk 1988), and which are listed in Table 13. Mineral quantities were determined by the image analysis system, by both the major and minor mineral routines. This involved measuring the relative areal fraction for each mineral or mineral group (e.g. gangue) within the polished section. The relative areal fraction (area %) is stereologically equivalent to volume %, the units in which the data are presented.

The major mineral routine determined the mineral quantities of the gangue, pyrite, pyrrhotite, chalcopyrite, sphalerite, arsenopyrite, freibergite and galena. The minor mineral routine re-analysed the mineral quantities of freibergite and galena with greater precision and accuracy, and additionally analyzed pyrargyrite, hessite, electrum and gudmundite. Because of the extremely trace abundances and relatively few measurements possible for these minor minerals in the various products (except for galena), the raw mineral quantities contain large uncertainties. Consequently, these data cannot be considered quantitative, but instead are of a semi-quantitative to qualitative nature.

The condensed raw modal analyses for the 17 mill products are presented in Table 14. The most striking feature of the table is that all of the minerals, except galena, determined by the minor mineral routine were undetected in many of the mill products i.e. no observations of pyrargyrite and hessite in the Zn-circuit concentrates. Furthermore, of those minerals observed in particular samples, the value of area % given often constitutes that of only one or two grains i.e. one grain of electrum in the Cu-scavenger tailings. Therefore, the raw values for the mineral quantities of the minor minerals only provide imprecise data on the relative magnitude of the abundances of these minerals. In addition to estimating the general path through the circuit of the various minerals of interest, it is also desirable to determine their absolute abundances. As the composition of each of these minerals has been determined (Healy & Petruk 1988), the absolute abundances of these minerals in the various products will permit mineral-metal accounting to be performed i.e. determination of the proportions of the element contents in each of the products attributable to each mineral.

The raw modal analyses were adjusted by material balance calculations in order to simulate their behaviour through the circuit under the steady state. For all the minerals measured by the major mineral routine, as well as galena, the raw data were inserted directly for data processing. For freibergite, electrum, pyrargyrite, gudmundite and hessite the raw data had to be manipulated in preparation for data processing. For example, null values (i.e.

TABLE 13

LIST OF ORE MINERALS OBSERVED IN THE TROUT LAKE ORES.

NAME	FORMULA	ABUNDANCE
THE COMMON SULPHIDES.		
Pyrite	FeS_2	Ubiquitous, Major/Minor.
Chalcopyrite	$CuFeS_2$	Ubiquitous, Major/Minor.
Sphalerite	ZnS	Ubiquitous, Major/Minor.
Pyrrhotite	$Fe_{(1-x)}S$	Common, Major/Trace.
Marcasite	FeS_2	Common, Major/Trace.
Galena	PbS	Common, Major/Trace.
Arsenopyrite	$FeAsS$	Common, Major/Trace.
Cubanite	$CuFe_2S_3$	Occasional, Trace.
Acanthite	Ag_2S	Rare, Trace.
THE OXIDES.		
Magnetite	Fe_3O_4	Occasional, Major/Minor.
Cassiterite	SnO_2	Rare, Trace.
THE INTERMETALLIC AND ALLOY-LIKE COMPOUNDS.		
Mercurian Electrum	Ag,Au,Hg	Common, Trace.
Dyscrasite	Ag_3Sb	Rare, Trace.
THE ANTIMONIAL SULFOSALTS.		
Freibergite	$(Cu,Ag)_{10}(Fe,Zn)_2(Sb,As)_4S_{13}$	Occasional, Minor/Trace.
Gudmundite	$FeSbS$	Occasional, Trace.
Pyrargyrite	Ag_3SbS_3	Rare, Trace.
Freieslebenite	$Ag_2Sb_2Pb_2S_6$	Rare, Trace.
Boulangerite	$Pb_5Sb_4S_{11}$	Rare, Trace.
Costibite	$CoSbS$	Rare, Trace.
Bournonite	$PbCuSbS_3$	Rare, Trace.
THE TELLURIDES.		
Hessite	Ag_2Te	Rare, Trace.
Rucklidgeite	$(Bi,Pb)_3Te_4$	Rare, Trace.
Pilsenite(?)	$(Bi,Pb)_8(Te,Se)_7$	Rare, Trace.
Volynskite	$BiAgTe_2$	Rare, Trace.
THE SELENIDES.		
Clausthalite	$PbSe$	Occasional, Minor/Trace.
Naumannite	Ag_2Se	Rare, Trace.

Presented in order of decreasing abundance of the mineral groups, and of the minerals in each group. From Healy & Petruk (1988).

undetected) were replaced by 0.00001 area %, a first approximation to the true value, which lies between the lowest detectable quantity and absolute zero area %.

Because the samples represent different, yet compositionally interdependent points in a circuit, the composite random error of the values for a given mineral in the 17 products is less than when each are viewed independently. MATBAL uses least squares refinement, whereby the sum of the squares of all adjustments is minimised, to adjust the experimental estimates or first approximations (Laguitton 1985). Therefore, despite the high random error in each value for a mineral in a product, the adjusted values that constitute a coherent set of data that approach the true values in a steady state operation. It should be stressed that MATBAL refines and adjusts each variable independently, such that cross-balancing is not neccessarily achieved, though ideally the closer the true values are approached for each variable the closer the cross-balance achieved.

Th adjusted modal analyses are given in Table 15, while the calculated recoveries are given in Table 16. It is apparent by comparing Tables 14 and 15 that the adjustments to the raw data are relatively minor for all the minerals measured by the major mineral routine, as well as galena, whereas those for freibergite, electrum, pyrargyrite and gudmundite are high. The average adjustment as a percentage of the raw value equals 4.2 % for gangue, 7.4 % for sphalerite and 24 % for galena, and principally reflects counting statistics, and biases in sample preparation and sample representivity. Calculated Cu and Zn assays derived from the adjusted modal analyses are 22 % Cu in the Cu-2nd cleaner concentrate and 43 % Zn in the Zn-2nd cleaner concentrate. These compare favourably with the adjusted chemical assays of 28.0 % Cu and 47.3 % Zn in the respective concentrates, especially in view of the small amount of material that was examined by image analysis compared to chemical analysis.

Even more favourable are the comparisons of 91.74 % recovery of Cu and 96.22 % recovery of chalcopyrite in the Cu-2nd cleaner concentrate, and 82.47 % recovery of Zn and 84.94 % recovery of sphalerite in the Zn-2nd cleaner concentrate. These results for chalcopyrite and sphalerite indicate that while the absolute abundances may be subject to sampling and analytical errors or biases, these are relatively consistent from sample to sample and yield accurate trends in the behaviour of minerals.

The mineral recoveries given in Table 16 are plotted as histograms in Figure 4, and graphically describe the behaviour of each of the minerals of interest through the circuit. Those minerals exhibiting similar behaviour are grouped vertically.

Group 1 consists of gangue, pyrite and pyrrhotite, all of which behave as essentially inert minerals with greater than 75 % recovery in the tailings. Pyrite exhibits 29 % recovery in the Cu-rougher concentrate, but is substantially rejected by the Cu-cleaning circuit. The bulk of the pyrite retained in the Cu-concentrate occurs in composite particles consisting of pyrite enveloped in chalcopyrite. Pyrrhotite exhibits 27 % recovery in the Zn-rougher concentrate, and is only partly rejected by the Zn-cleaning circuit, down to 18.5 % in the Zn-2nd cleaner concentrate. Petrographic examination reveals that this pyrrhotite consists largely of diablastic-textured particles

TABLE 14

RAW MODAL ANALYSES (BY VOL %) OF THE MINERALS OF INTEREST IN THE TROUT LAKE MILL PRODUCTS.

PRODUCT.	MODAL QUANTITIES FROM THE MAJOR MINERAL. ROUTINE.						MODAL QUANTITIES FROM THE MINOR MINERAL ROUTINE.					
	Gang	Py	Po	Cpy	Sphal	Aspy	Gal	Freib	Elect	Pyrarg	Gudmun	Hess
1. Feed.	84.0	9.17	0.33	3.46	2.82	0.09	0.046	0.00004	0.00024	-----	0.00006	-----
2. Cu Scav. Conc.	22.1	37.6	1.21	23.4	15.0	0.57	0.019	0.00021	0.00005	-----	0.00038	-----
3. Cu Roug. Conc.	26.9	20.0	0.78	43.1	8.64	0.40	0.036	0.00228	0.00035	0.00011	0.00028	0.00140
4. Cu 1st Cl. Fd.	19.4	23.4	0.20	45.8	10.9	0.13	0.033	0.00042	-----	0.00024	-----	-----
5. Cu 1st Cl. Conc.	7.53	15.3	1.44	65.5	10.1	0.08	0.044	0.01545	0.00079	0.00007	0.00030	0.00121
6. Cu 2nd Cl. Conc.	6.24	7.90	0.59	77.5	7.67	0.07	0.071	0.01361	0.01370	0.00315	-----	0.00039
7. Zn Roug. Fd.	84.0	10.9	0.32	0.16	4.29	0.22	0.036	-----	-----	-----	-----	-----
8. Zn Roug.Conc.	21.4	18.3	1.41	2.63	55.4	0.55	0.175	0.00011	0.00013	-----	0.00035	-----
9. Zn 1st Cl. Conc.	13.8	26.0	2.64	2.55	54.1	0.28	0.239	-----	0.00005	-----	0.00008	-----
10. Zn 2nd Cl. Conc.	6.43	10.7	2.01	1.56	78.8	0.39	0.399	0.00030	0.00018	-----	0.00027	-----
11. Cu Roug. Tail.	83.6	11.1	0.61	0.48	3.82	0.31	0.021	-----	-----	-----	0.00020	-----
12. Cu Scav. Tail.	86.4	8.83	0.58	0.50	3.61	0.02	0.045	0.00005	0.00003	0.00006	0.00025	-----
13. Cu 1st Cl. Tail.	35.1	41.5	0.40	10.5	12.2	0.31	0.047	0.00964	0.00099	-----	0.00038	0.00019
14. Cu 2nd Cl. Tail.	23.5	33.7	0.65	26.9	14.9	0.35	0.094	0.00446	0.00064	0.00014	0.00004	0.00056
15. Zn 1st Cl. Tail.	52.9	42.0	2.28	0.60	1.44	0.43	0.137	0.00041	-----	-----	0.00024	-----
16. Zn 2nd Cl. Tail.	30.2	57.5	2.38	3.26	4.85	0.72	0.453	0.00015	0.00009	-----	0.00014	0.00006
17. Zn Roug. Tail.	88.7	10.0	0.94	0.11	0.03	0.18	0.009	-----	-----	0.00126	0.00001	-----

'-----' denotes not detected.

TABLE 15

MATERIAL BALANCE ADJUSTED MODAL ANALYSES OF THE MINERALS OF INTEREST IN THE TROUT LAKE MILL PRODUCTS.

PRODUCT.	MODAL QUANTITIES FROM THE MAJOR MINERAL. ROUTINE.						MODAL QUANTITIES FROM THE MINOR MINERAL ROUTINE.					
	Gang	Py	Po	Cpy	Sphal	Aspy	Gal	Freib	Elect	Pyrarg	Gudmun	Hess
1. Feed.	80.37	9.61	0.40	4.60	3.82	0.03	0.024	0.00021	0.00011	0.00014	0.00011	0.00004
2. Cu Scav. Conc.	22.10	37.80	1.23	19.06	15.02	0.59	0.019	0.00020	0.00005	0.00009	0.00393	0.00001
3. Cu Roug. Conc.	20.46	21.95	0.30	39.50	9.59	0.16	0.039	0.00216	0.00120	0.00047	0.00009	0.00034
4. Cu 1st Cl. Fd.	20.62	22.81	0.32	38.74	9.95	0.17	0.042	0.00232	0.00117	0.00052	0.00011	0.00035
5. Cu 1st Cl. Conc.	7.93	11.53	0.37	58.81	8.58	0.09	0.046	0.00306	0.00129	0.00083	0.00013	0.00046
6. Cu 2nd Cl. Conc.	6.13	8.64	0.34	62.60	7.77	0.06	0.041	0.00286	0.00137	0.00076	0.00010	0.00044
7. Zn Roug. Fd.	85.41	10.28	0.44	0.20	3.48	0.04	0.025	0.00001	0.00001	0.00009	0.00015	0.00001
8. Zn Roug.Conc.	20.36	19.20	1.74	1.48	51.84	0.35	0.227	0.00007	0.00006	0.00010	0.00128	0.00001
9. Zn 1st Cl. Conc.	14.29	26.33	1.97	2.36	49.23	0.37	0.303	0.00003	0.00007	0.00010	0.00102	0.00001
10. Zn 2nd Cl. Conc.	6.43	10.42	1.66	1.83	72.29	0.30	0.267	0.00001	0.00007	0.00010	0.00094	0.00001
11. Cu Roug. Tail.	84.71	10.25	0.42	0.57	3.75	0.04	0.023	0.00001	0.00001	0.00009	0.00019	0.00001
12. Cu Scav. Tail.	86.09	9.68	0.41	0.19	3.52	0.03	0.023	0.00001	0.00001	0.00009	0.00011	0.00001
13. Cu 1st Cl. Tail.	38.31	38.53	0.26	10.75	11.86	0.28	0.037	0.00128	0.00099	0.00010	0.00008	0.00021
14. Cu 2nd Cl. Tail.	22.98	35.73	0.34	27.17	15.37	0.31	0.088	0.00473	0.00064	0.00139	0.00040	0.00059
15. Zn 1st Cl. Tail.	54.72	40.82	1.92	0.61	1.44	0.48	0.131	0.00023	0.00001	0.00010	0.00212	0.00001
16. Zn 2nd Cl. Tail.	29.41	56.94	2.57	3.37	4.86	0.51	0.374	0.00006	0.00007	0.00010	0.00117	0.00002
17. Zn Roug. Tail.	90.04	9.64	0.34	0.10	0.03	0.01	0.011	0.00001	0.00001	0.00009	0.00007	0.00001

TABLE 16

MATERIAL BALANCE CALCULATED RECOVERIES OF THE MINERALS OF INTEREST IN THE TROUT LAKE MILL PRODUCTS.

PRODUCT.	MAJOR MINERAL ROUTINE.						MINOR MINERAL ROUTINE.					
	Gang	Py	Po	Cpy	Sphal	Aspy	Gal	Freib	Elect	Pyrarg	Gudmun	Hess
1. Feed.	100.00	100.00	100.00	100.00	100.00	100.00	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu Scav. Conc.	0.53	7.58	5.87	7.99	7.58	37.69	1.5	1.8	0.9	1.5	67.1	0.5
3. Cu Roug. Conc.	3.24	29.12	9.59	109.49	31.99	67.86	20.4	131.1	142.2	43.4	10.3	107.1
4. Cu 1st Cl. Fd.	3.49	32.26	10.84	114.49	35.39	76.61	23.5	150.2	147.2	51.9	13.1	119.5
5. Cu 1st Cl. Conc.	0.78	9.50	7.24	101.22	17.77	23.72	14.9	115.6	94.9	47.6	9.3	89.9
6. Cu 2nd Cl. Conc.	0.54	6.35	5.99	96.22	14.37	14.97	11.9	96.5	90.0	39.1	6.3	77.5
7. Zn Roug. Fd.	100.70	101.38	102.72	4.02	86.32	113.98	97.9	5.5	10.2	62.3	127.9	23.0
8. Zn Roug.Conc.	1.60	12.60	27.22	2.03	85.63	74.02	58.7	2.2	3.3	4.6	71.7	1.4
9. Zn 1st Cl. Conc.	1.21	18.69	33.41	3.50	87.91	84.95	84.8	0.8	4.5	5.0	61.6	2.2
10. Zn 2nd Cl. Conc.	0.36	4.87	18.52	1.79	84.94	45.07	49.0	0.1	3.1	3.3	37.5	1.0
11. Cu Roug. Tail.	99.99	101.22	99.88	11.77	93.21	122.72	89.6	5.3	10.9	62.3	160.8	23.0
12. Cu Scav. Tail.	99.46	93.65	94.01	3.78	85.63	85.03	88.1	3.5	10.0	60.9	93.7	22.5
13. Cu 1st Cl. Tail.	2.71	22.76	3.60	13.27	17.62	52.89	8.6	34.6	52.2	4.3	3.9	29.6
14. Cu 2nd Cl. Tail.	0.24	3.14	1.25	4.99	3.40	8.75	3.1	19.1	5.0	8.5	3.0	12.4
15. Zn 1st Cl. Tail.	1.24	7.73	8.71	0.24	0.69	28.95	9.7	2.0	0.2	1.3	34.2	0.5
16. Zn 2nd Cl. Tail.	0.85	13.82	14.89	1.71	2.97	39.51	35.8	0.7	1.4	1.7	24.1	1.3
17. Zn Roug. Tail.	99.10	88.78	75.49	1.99	0.69	39.96	39.1	3.4	7.0	57.7	56.2	21.5

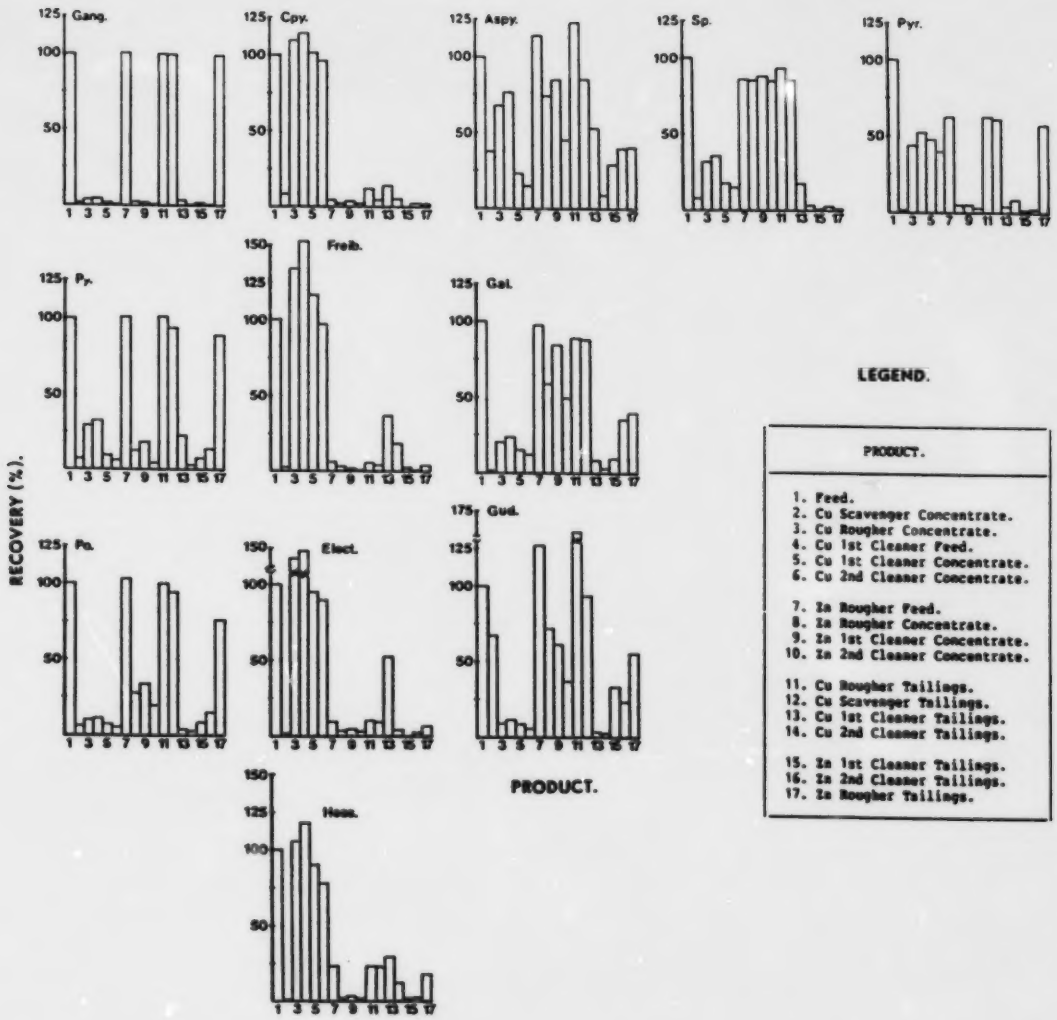

FIGURE 4.
THE DISTRIBUTION OF THE RECOVERIES CALCULATED BY MATBAL FOR THE MINERALS OF INTEREST IN THE 17 MILL PRODUCTS.

of pyrrhotite and sphalerite, as well as pyrrhotite rods and blebs in sphalerite.

Group 2 consists of chalcopyrite, freibergite, electrum and hessite, all of which float readily into the Cu-circuit concentrates with recoveries in the 90's, except for hessite, in the Cu-2nd cleaner concentrate. Freibergite, electrum and hessite differ from chalcopyrite in that they exhibit high rejection in the Cu-cleaning circuits with resultant high circulating loads of these minerals. The high circulating loads may be attributable to the fine-grain size and hence liberation, or conversely coarse-grain size coupled with high specific gravity, or flotation characteristics related to the Minerec collector, or a combination of these.

Group 3 consists of arsenopyrite, galena and gudmundite, all of which exhibit low recoveries in the Cu-2nd cleaner concentrate, but moderately high recoveries in both the Zn-2nd cleaner concentrates and the tailings. Arsenopyrite exhibits a moderately high recovery of 67.9 % in the Cu-rougher concentrate, while gudmundite exhibits a similar recovery of 67.1 % in the Cu-scavenger concentrate. However, both these minerals are largely rejected as they pass through Cu-circuit cleaning. These minerals are interpreted to be largely inert, but whose behaviour is modified by the host minerals in the composite particles in which it may occur. On the basis of textural habit, arsenopyrite, galena and gudmundite were all predicted to partly occur in composite particles with sphalerite (Healy & Petruk 1988), and are thus expected to partly report to the Zn-2nd cleaner concentrate. Free grains and particles consisting predominantly of any of these three minerals are expected to report to the tailings. Both arsenopyrite and galena are expected to occur partly in particles with pyrite, which would also report to the tailings. The apparently inert behaviour of galena is in contrast to the common practise of initially floating a Pb-Cu bulk concentrate in many processing plants of massive sulfide ores (Petruk 1986). However, in such plants where the Pb grade and/or the associated Ag grade justifies recovery, reagent mixtures, specifically different collectors, are used for galena recovery.

Group 4 consists of sphalerite only, which is almost totally recovered in the circuit; principally (84.94 %) in the Zn-2nd cleaner concentrate, and secondarily a significant recovery (14.37 %) in the Cu-2nd cleaner concentrate. The similarity of the galena distribution to that of sphalerite (except for significant losses of galena to the tailings) further suggests that the inert galena occurs principally in composite particles containing sphalerite as the dominant host. It is also noteworthy that the magnitude of the pyrite and sphalerite recoveries in the Cu-rougher concentrate are comparable, but that through the Cu-cleaning circuit the pyrite exhibits higher rejection than the sphalerite.

Group 5 consists of pyrargyrite only, which is principally lost to the tailings (57.7 %), but with high recoveries (39.1 %) in the Cu-2nd cleaner concentrate, and unexpectedly low recoveries (3.3 %) in the Zn-2nd cleaner concentrate. Healy & Petruk (1988) had predicted that the pyrargyrite, which predominantly occurs along with pyrrhotite as fine-grained inclusions in galena interstitial to sphalerite, would partly report to the Zn-2nd cleaner concentrate as sphalerite-galena-pyrargyrite particles.

CONCENTRATION FACTORS

Concentration factors (as defined in CHEMICAL ANALYSIS) were also calculated for the adjusted mineral quantities of the 12 minerals of interest. This was performed to directly compare the relative concentration of the minerals in the various products. The Fortran 77 program 'CONFAC' performed the calculations of the concentration factors. The concentration factors for 12 minerals in the 17 mill products are given in Table 17. These data were plotted as discontinuous curves in Figure 5. The results yield the same general pattern of 5 relatively coherent mineral groups as that indicated by the plot of recoveries in Figure 4. However, in that the concentration of each of the minerals is compared against one another, the preferential concentration or depletion of the minerals is more clearly revealed.

The Cu-Circuit : The Cu-rougher produces relative concentration of chalcopyrite, pyrite, sphalerite, freibergite, electrum, arsenopyrite and pyrargyrite, while the Cu-scavenger produces relative depletion in gangue, electrum and pyrargyrite. Pyrite, pyrrhotite, sphalerite, arsenopyrite and gudmundite exhibit their highest relative concentration (in the Cu-circuit) in the Cu-scavenger i.e. higher cell tailings weir height to recover both the coarse-grained free chalcopyrite and middling particles containing chalcopyrite. The latter particles would likely contribute to the increased recovery of locked pyrite, pyrrhotite and sphalerite particularly, while free grains of all 5 of these minerals would also be partly recovered due to the implied lower selectivity of this unit. These 5 minerals with the exception of sphalerite are subsequently largely rejected through the Cu-rougher and Cu-cleaning stages. In contrast, the sphalerite, though relatively less enriched in the Cu-rougher, is not preferentially rejected during Cu-cleaning, and exhibits virtually the identical concentration factor in the Cu-2nd cleaner concentrate as that of the Cu-scavenger concentrate i.e. 3.42 and 3.58, respectively.

The relative depletion of electrum and the relative concentration of both pyrite and arsenopyrite in the Cu-scavenger is interesting in terms of the behaviour of Au. Au exhibits its highest relative concentration in the Cu-scavenger, while electrum the interpreted dominant Au-carrier exhibits relative depletion. Because of the uncertainty in the adjustments of the raw modal analyses for electrum (i.e. poor counting statistics) it is possible that this value is erroneously low and the value for the Cu-rougher concentrate, correspondingly high. Nonetheless, it may suggest that at least a significant fraction of the Au occurs in pyrite and possibly arsenopyrite as 'invisible' Au. However, in as much as the concentration factor[3] for Au is far in excess of that for electrum, yet the invisible Au content of pyrite and/or arsenopyrite is interpreted to constitute only 10 to 15 % of the Au in the ore, the former conclusion is more probable. Similarly, the poor counting statistics and adjustments on such values of high uncertainty may largely explain the high concentration factor of 11.99 % (and hence high circulating load) for freibergite in the Cu-2nd cleaner tailings. However, as Ag exhibits a concentration factor of 2.57 in Cu-2nd cleaner tailings, and freibergite is a significant Ag-carrier, this preferential rejection of freibergite into the

[3] Concentration factors for different datasets are inequivalent and can be compared in a qualitative manner only.

Cu-2nd cleaner tailings is probably true.

Gangue is the only mineral (group) that exhibits relative depletion in the Cu-2nd cleaner concentrate. The modal amounts of gangue drop from 80.4 % in the feed to 6.13 % in this concentrate, and thus alone elevates the proportions of all other minerals relatively. As the enrichment ratio for chalcopyrite in the Cu-2nd cleaner concentrate is only 13.6, the rejection of 99.46 % of the gangue from the Cu-circuit, will alone generate an enrichment ratio of 4 for chalcopyrite. Thus, the relative depletion of gangue alone in the Cu-2nd cleaner concentrate and accompanying moderate relative concentration of most other minerals, reflects the efficiency of the Cu-circuit rejection of gangue, and its limited selectivity in sulfide rejection. For example, the enrichment ratios for pyrite, pyrrhotite, sphalerite and galena in the Cu-2nd cleaner concentrate are 0.90, 0.85, 2.03 and 1.71, respectively. The grade of sphalerite in the Cu-concentrate is therefore twice that of the feed, and thus also twice the nominally acceptable[4] level. It is noteworthy that the Cu-cleaning circuit does not reject significant amounts of gangue, pyrite or sphalerite i.e. exhibit relative depletion or low relative concentration in the Cu-1st and 2nd cleaner tailings. In that the behaviour of pyrite and sphalerite are similar in the Cu-circuit, argues (at this juncture) against activation as the cause of the high sphalerite lost to the Cu-2nd cleaner concentrate. However, the above similarities are a generalisation in that sphalerite does undergo moderate relative concentration in the Cu-cleaning circuit, while pyrite undergoes moderate relative depletion.

The Zn-Circuit : Sphalerite, galena arsenopyrite, gudmundite, pyrrhotite, pyrite, pyrargyrite, electrum and chalcopyrite in that order of magnitude, exhibit relative concentration in the Zn-2nd cleaner concentrate, while hessite, gangue and freibergite exhibit relative depletion. The concentration factor for sphalerite in the Zn-concentrate is 70.19, and substantially exceeds that of galena (41.26) and arsenopyrite (37.09) in this concentrate, and gudmundite (47.73) in the Zn-1st cleaner tailings. This high concentration factor for sphalerite reflects the enrichment ratio of 18.9, which despite higher analytical uncertainties is not significantly different from that of 18.4 for Zn assays. This reflects the excellent recovery and selective flotation of sphalerite in the Zn-circuit. Both pyrite and pyrrhotite exhibit concentration factors of > 4, while galena, arsenopyrite and gudmundite exhibit concentration factors of > 30 in the Zn-concentrate. These minerals are unlikely to be affected by activation with $CuSO_4$, and thus to be floated preferentially into the Zn-circuit as opposed to the Cu-circuit, implies that they are essentially inert, but whose flotation behaviour is partly modified by the degree of liberation and locking with sphalerite. Petrographic observation indicates that at least the bulk of the pyrite, pyrrhotite, arsenopyrite and galena in the Zn-concentrate occurs in composite particles consisting largely of sphalerite.

Because the gangue is largely rejected initially in the Zn-rougher, reporting directly to the tailings, and chalcopyrite, electrum, freibergite and hessite have already been largely removed in the Cu-circuit, the

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[4] According to a widely held Mineral Processor's Rule of Thumb pertaining to the processing of base metal sulfide ores, 'an acceptable Zn grade in the Cu-concentrate should not exceed that of the feed'.

concentration factors for pyrite, pyrrhotite, arsenopyrite, galena and gudmundite are high in the Zn-cleaning circuit. Nonetheless, it is apparent from the concentration factors of these latter minerals in the Zn-1st and 2nd cleaner tailings, that while the Zn-rougher exhibits moderate selectivity in sulfide flotation, the Zn-1st and 2nd cleaners yield greater rejection. This is a desirable configuration, in that having been rejected by the Zn-rougher, the lack of a Zn-scavenger results in the loss of the mineral to the tailings without further opportunity for recovery. Thus, while the gangue, pyrite, pyrrhotite and pyrargyrite report principally to the Zn-rougher tailings, with moderate circulating loads of both pyrite and pyrrhotite, the galena, arsenopyrite and gudmundite though partly rejected during Zn-cleaning report largely to the Zn-concentrate.

The Zn-rougher tailings exhibit relative depletion for chalcopyrite and sphalerite and freibergite only, exhibiting at least moderate relative concentration for all other minerals. Gangue, pyrrhotite, pyrite, pyrargyrite and gudmundite are the only minerals to exhibit high (> 10) relative concentration in the Zn-rougher tailings, which with the exception of pyrargyrite (Ag-bearing) is highly desirable. The behaviour of pyrargyrite is ambivalent and may reflect free pyrargyrite reporting to the Cu-concentrate, with the remainder which is expected to be principally as fine-grained inclusions in galena and/or pyrrhotite (Healy & Petruk 1988) reporting to the tailings. An additional complicating factor is that both bournonite and boulangerite were misidentified (at least occasionally) as pyrargyrite during the EDS-assisted image analysis (minor mineral routine), and thus partly distorts the data. The pyrrhotite reporting to the Zn-concentrate was petrographically observed as diablastic (Healy & Petruk 1988) pyrrhotite in sphalerite, and rods and blebs of pyrrhotite in sphalerite.

Chalcopyrite generally exhibits relative depletion in the Zn-circuit concentrates, though the concentration factors consistently increase through Zn-cleaning to give relative concentration (i.e. CF = 1.48) in the Zn-concentrate. Petrographic examination revealed four principal forms of chalcopyrite in the Zn-concentrate : (1) diablastic chalcopyrite in sphalerite (See Plate 1A); (2) chalcopyrite-disease in sphalerite (See Plate 1B); (3) rare, coarse-grained (or 'over-sized'), free and sub-free chalcopyrite grains (See Plate 2A and 2B); and (4) middling particles of pyrite-chalcopyrite and gangue-chalcopyrite (See Plate 2C and 2D). The average equivalent square diameter, determined by optical measurements, of twenty of the largest grains of free and sub-free chalcopyrite is 130 um, with the largest grain measuring 440 x 150 um. Unfortunately, few of these coarse-grained, essentially free chalcopyrite grains were included in the area of the sample surface traversed during image analysis, and thus the data is not completely representative.

Possibly significant is the apparent frequency of sphalerite grains containing chalcopyrite-disease. Based solely on petrographic examination, sphalerite grains reporting to the Cu-concentrate apparently have a higher frequency of chalcopyrite-disease than those reporting to the Zn-concentrate. As chalcopyrite-disease generally contributes less than 5 % of the volume of a given sphalerite grain, it would not be expected to substantially affect the flotation behaviour of the sphalerite. However, during grinding the sphalerite tends to break along fractures, cleavages and twin planes, where chalcopyrite-disease is concentrated, and the blebs and rods of chalcopyrite may not be

completely liberated, but remain attached. These may in this case constitute a greater area % of the surface of the grain in proportion to their volume % of the sphalerite grain. The relatively common observation of chalcopyrite blebs attached on essentially straight sphalerite grain edges, suggests that this is true for a small, yet significant number of sphalerite grains, and may affect the flotation of these sphalerite grains. These observations are generally consistent with those of Petruk & Mainwaring (1985), who observed that approximately 10 % of the chalcopyrite-disease is retained in the ground sphalerite, the bulk being liberated when the sphalerite breaks along the planes in which it is concentrated. This is considered true for Trout Lake sphalerite, but that in being virtually ubiquitous in the unbroken ore, sphalerite containing persistent chalcopyrite-disease may still constitute a significant portion of the ground sphalerite. However, as the bulk of sphalerite reports to the Zn-concentrate, and as persistent chalcopyrite-disease is perhaps twice as abundant in the Cu-concentrate as in the Zn-concentrate, the bulk of the persistent chalcopyrite-disease reports to the Zn-concentrate.

TABLE 17

CONCENTRATION FACTORS FOR MINERAL QUANTITIIES IN THE 17 MILL PRODUCTS.

PRODUCT.	MINERAL.											
	Gang	Py	Po	Cpy	Sphal	Aspy	Gal	Freib	Elect	Pyrarg	Gud	Hess
1. Feed.	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
2. Cu Scav Conc.	0.34	4.87	3.81	5.13	4.87	24.35	0.98	1.18	0.56	0.80	44.24	0.31
3. Cu Roug Conc.	0.19	1.68	0.55	6.33	1.85	3.93	1.20	7.58	8.04	2.47	0.60	6.26
4. Cu 1st Cl Feed.	0.18	1.66	0.56	5.89	1.82	3.96	1.22	7.72	7.43	2.60	0.70	6.12
5. Cu 1st Cl Conc.	0.12	1.47	1.13	15.64	2.75	3.67	2.35	17.83	14.35	7.25	1.45	14.07
6. Cu 2nd Cl Conc.	0.12	1.39	1.32	21.07	3.15	3.10	2.64	21.09	19.28	8.40	1.41	17.03
7. Zn Roug Feed.	5.92	5.96	6.13	0.24	5.08	7.43	5.81	0.27	0.51	3.58	7.60	1.39
8. Zn Roug Conc.	0.39	3.06	6.66	0.49	20.77	17.86	14.48	0.51	0.84	1.09	17.81	0.38
9. Zn 1st Cl Conc.	0.33	5.13	9.22	0.96	24.12	23.08	23.63	0.27	1.19	1.34	17.35	0.47
10. Zn 2nd Cl Conc.	0.30	4.02	15.39	1.48	70.19	37.09	41.26	0.18	2.36	2.65	31.69	0.93
11. Cu Roug Tails.	4.55	4.61	4.54	0.54	4.24	5.76	4.14	0.21	0.39	2.78	7.46	1.08
12. Cu Scav Tails.	6.14	5.77	5.88	0.24	5.28	5.73	5.49	0.27	0.52	3.69	5.73	1.43
13. Cu 1st Cl Tails.	0.34	2.89	0.47	1.68	2.24	6.72	1.11	4.39	6.48	0.51	0.52	3.60
14. Cu 2nd Cl Tails.	0.15	1.98	0.45	3.15	2.14	5.50	1.95	11.99	3.10	5.29	1.94	7.85
15. Zn 1st Cl Tails.	1.69	10.52	11.89	0.33	0.93	39.62	13.52	2.71	0.23	1.77	47.73	0.62
16. Zn 2nd Cl Tails.	0.42	6.86	7.44	0.85	1.47	19.69	18.05	0.33	0.74	0.83	12.32	0.58
17. Zn Roug Tails.	20.61	18.45	15.63	0.40	0.14	6.13	8.43	0.88	1.67	11.82	11.70	4.60

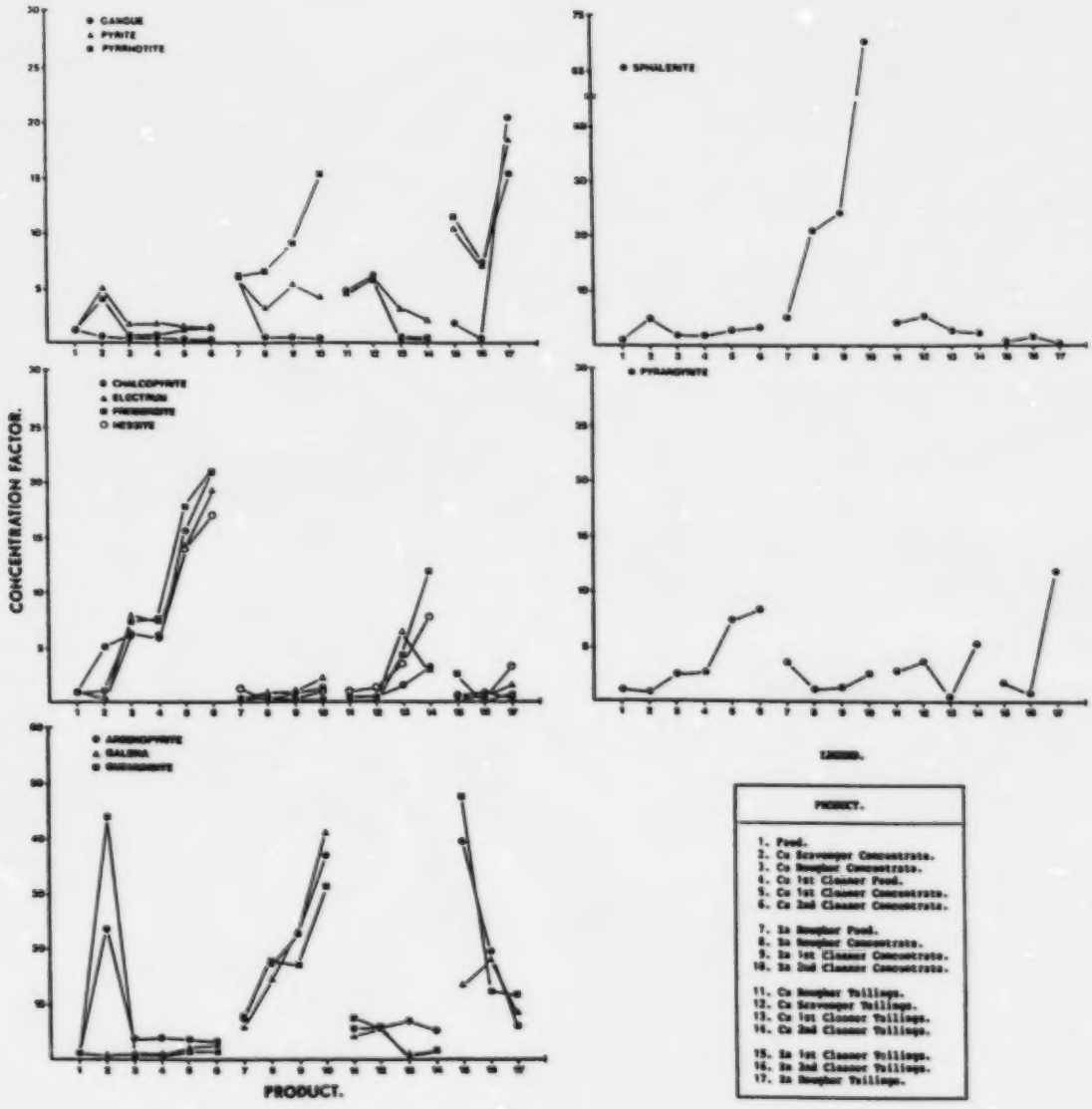

FIGURE 5.
THE CONCENTRATION FACTORS CURVES FOR MINERAL QUANTITIES OF THE MINERALS IN THE 17 MILL PRODUCTS.

FLOTATION EFFICIENCY

The use of concentration factors to determine the relative concentration of minerals is limited by the fact that disproportionate (relative to their proportions in the float feed) amounts of each mineral enter a flotation unit[5] at which the relative concentration is being determined. Thus, because 99.5 % of the gangue is lost to the Cu-scavenger tailings, and hence Cu-circuit, it elevates the relative concentration of all other minerals. If gangue had been excluded from the dataset many of the minerals would have exhibited relative depletion in the Cu-concentrate. Therefore, this is an arbitary, qualitative, comparative technique of evaluating the flotation behaviour of minerals. Furthermore, the method does not allow comparison of mineral behaviour at a given unit independently of the preceeding flotation units which have altered the proportions of the minerals entering the unit under study. In order to assess the intrinsic recovery at each unit, the recovery as a percent of the feed to that unit was calculated for each unit for several minerals of interest. This flotation efficiency expressed in percent is therefore in standardised units that are independent of the behaviour of other minerals. The flotation efficiencies for chalcopyrite, sphalerite, pyrite and gangue are given in Table 18, and graphically presented in Figure 6.

The Cu-Circuit : Despite the relative depletion of Cu and the moderate relative concentration of chalcopyrite in the Cu-scavenger, as determined by concentration factors, 68 % of the chalcopyrite entering the Cu-scavenger is recovered. The efficiency of chalcopyrite flotation is 90.3 %, 88.5 % and 95.1 % in the Cu-rougher and Cu-1st and 2nd cleaners, respectively. Incorporating the small circulating loads of chalcopyrite, results in a final recovery of 96.22 % for chalcopyrite in the Cu-concentrate.

The most notable result is that in the Cu-rougher and Cu-scavenger units, pyrite behaves similarly to sphalerite, while during Cu-cleaning the behaviour of pyrite resembles gangue. The efficiency of sphalerite and pyrite flotation are approximately 8 % in the Cu-scavenger, and approximately 23 % in the Cu-rougher. During Cu-cleaning there is a dichotomy in the flotation efficiencies for sphalerite and pyrite, with 50 % and 30 %, respectively in the Cu-1st cleaner and 81 % and 67 %, respectively in the Cu-2nd cleaner. Thus, while equivalently low fractions of pyrite and sphalerite and significantly lower fractions of gangue are recovered initially, Cu-cleaning rejects a greater proportion of the pyrite and gangue. Petrographic examination revealed that gangue retained in the Cu-concentrate consists principally of particles containing gangue minerals locked with chalcopyrite. In contrast, image analysis shows that only 22 % of the pyrite in the Cu-concentrate is locked with chalcopyrite, and of which approximately 70 % is sub-free, and should therefore behave as free grains of pyrite i.e. behave inertly (See appendix F). The sphalerite exhibits a steady increase in the flotation efficiencies through each unit of the Cu-circuit from 8.13 % to 25.6 % to 50.2 % to 80.9 %. Thus, Cu-cleaning rejects a progressively smaller fraction of the sphalerite, and appears to concentrate a fraction of the sphalerite that readily floats.

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[5] A flotation unit is defined here as one or more banks of flotation cells whose operating parameters are uniformly adjusted for a specific mineral separation, e.g. the Zn-rougher unit consists of five banks of flotation cells.

The abundance of free and sub-free sphalerite in the Cu-concentrate as observed petrographically, does not indicate any preferential enrichment of either middling particles of sphalerite or sub-free grains of sphalerite in the retained fraction.

The Zn-Circuit : The flotation efficiencies for sphalerite varies from 99.2 % in both the Zn-rougher and Zn-1st cleaner to 96.6 % in the Zn-2nd cleaner. Despite only 84.9 % of the sphalerite in the feed being recovered in the Zn-concentrate, 99.2 % of the sphalerite that enters the Zn-circuit is recovered in the Zn-concentrate. Thus, the Zn-circuit is operating at a very high level of efficiency for the recovery of sphalerite.

The Zn-rougher exhibits low efficiency for both pyrite and gangue flotation. The recovered component is partly rejected in the Zn-1st cleaner, and largely rejected in the Zn-2nd cleaner. Chalcopyrite exhibits a comparable pattern but at significantly higher levels of efficiency. All four minerals exhibit higher rejection in the Zn-2nd cleaner than in the Zn-1st cleaner, which with the exception of sphalerite results in significant circulating loads of these minerals, at least as a proportion of the amount initially recovered in the Zn-rougher. The higher flotation efficiencies of chalcopyrite relative to that of pyrite and gangue most probably reflect : (1) common middling particles of chalcopyrite-sphalerite which failed to float prior to Cu-activation; (2) chalcopyrite-pyrite and chalcopyrite-gangue middling particles that on probability simply failed to float initially in the Cu-circuit, due to marginal flotability: and (3) coarse-grained chalcopyrite which also is only marginally flotable. The latter two categories may exhibit higher recoveries in the Zn-circuit due to longer conditioning time, (but not because of Cu-activation), and the apparently higher overall flotation efficiency of the Zn-circuit.

TABLE 18

FLOTATION EFFICIENCIES FOR CHALCOPYRITE, SPHALERITE, PYRITE AND GANGUE.

PRODUCT.	Chalcopyrite Recovery			Sphalerite Recovery			Pyrite Recovery			Gangue Recovery		
	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.
Cu Scavenger.	11.77	7.99	67.9	93.21	7.58	8.13	101.22	7.58	7.49	99.99	0.53	0.53
Cu Rougher.	121.26	109.49	90.3	125.20	31.99	25.6	130.34	29.12	22.3	103.24	3.24	3.14
Cu 1st Cleaner.	114.49	101.22	88.5	35.39	17.77	50.2	32.26	9.50	29.5	3.49	0.78	22.4
Cu 2nd Cleaner.	101.22	98.22	95.1	17.77	14.37	80.9	9.50	6.35	66.8	0.78	0.54	69.2
Zn Rougher.	4.02	2.03	50.5	86.32	85.63	99.2	101.38	12.60	12.4	100.70	1.60	1.59
Zn 1st Cleaner.	3.74	3.50	93.6	88.60	87.91	99.2	26.42	18.69	70.7	2.45	1.21	49.4
Zn 2nd Cleaner.	3.50	1.79	51.1	87.91	84.94	96.6	18.69	4.87	26.1	1.21	0.26	29.8

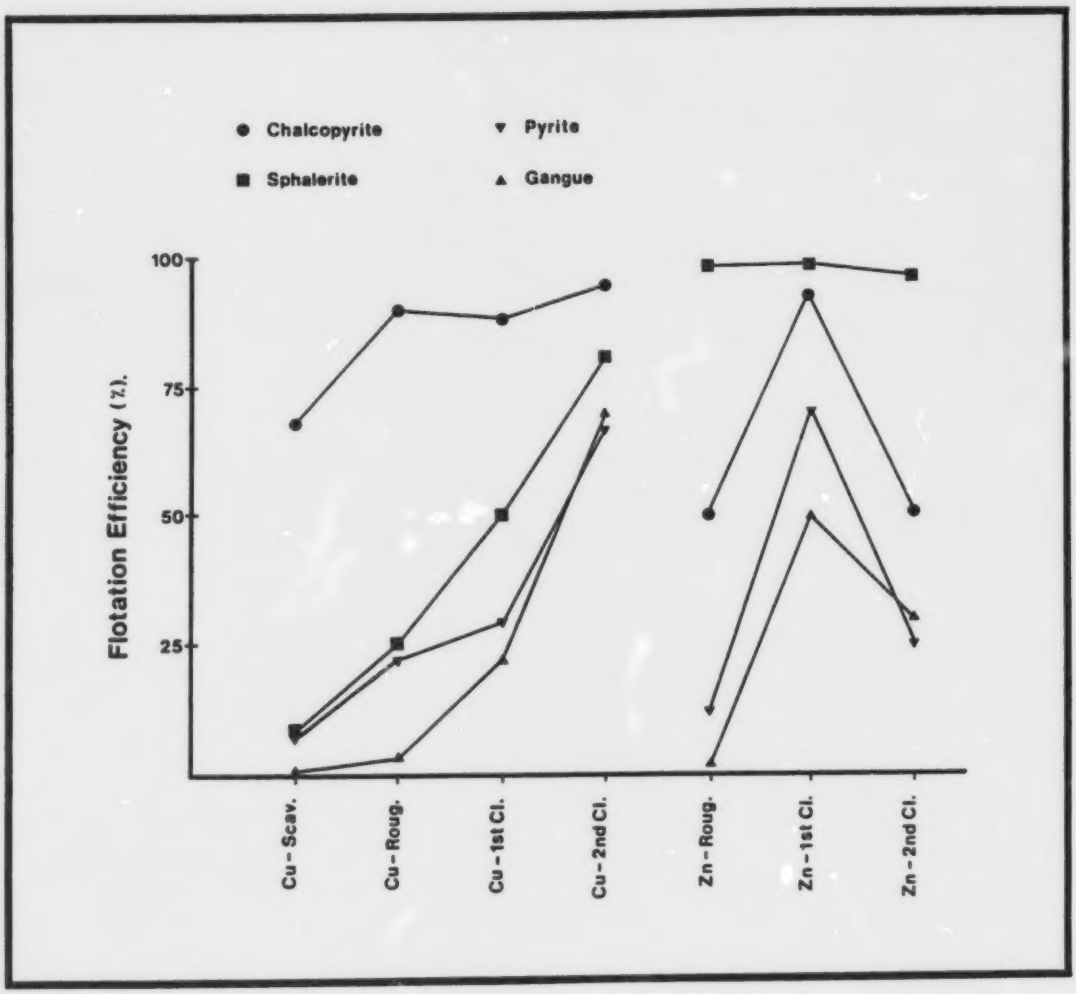


FIGURE 6.
PLOT OF FLOTATION EFFICIENCIES FOR CHALCOPYRITE, SPHALERITE, PYRITE AND GANGUE.

PLATE 1.

PLATE 1.

(A) Zn-CONCENTRATE : Myrmekitic chalcopyrite blebs in sphalerite, formed by diablastic growth during and after shearing. This is characteristic of, but not exclusive to the No. 2 lens of the North Zone. Accounts for significant losses of Zn to the Cu-concentrate, and Cu to the Zn-concentrate. Oil Immersion (OI). Plane Polarised Light (PPL).

(B) Zn-CONCENTRATE : Persistent chalcopyrite-disease consisting of fine-grained crystallographically oriented chalcopyrite rods and blebs in sphalerite. OI. PPL.

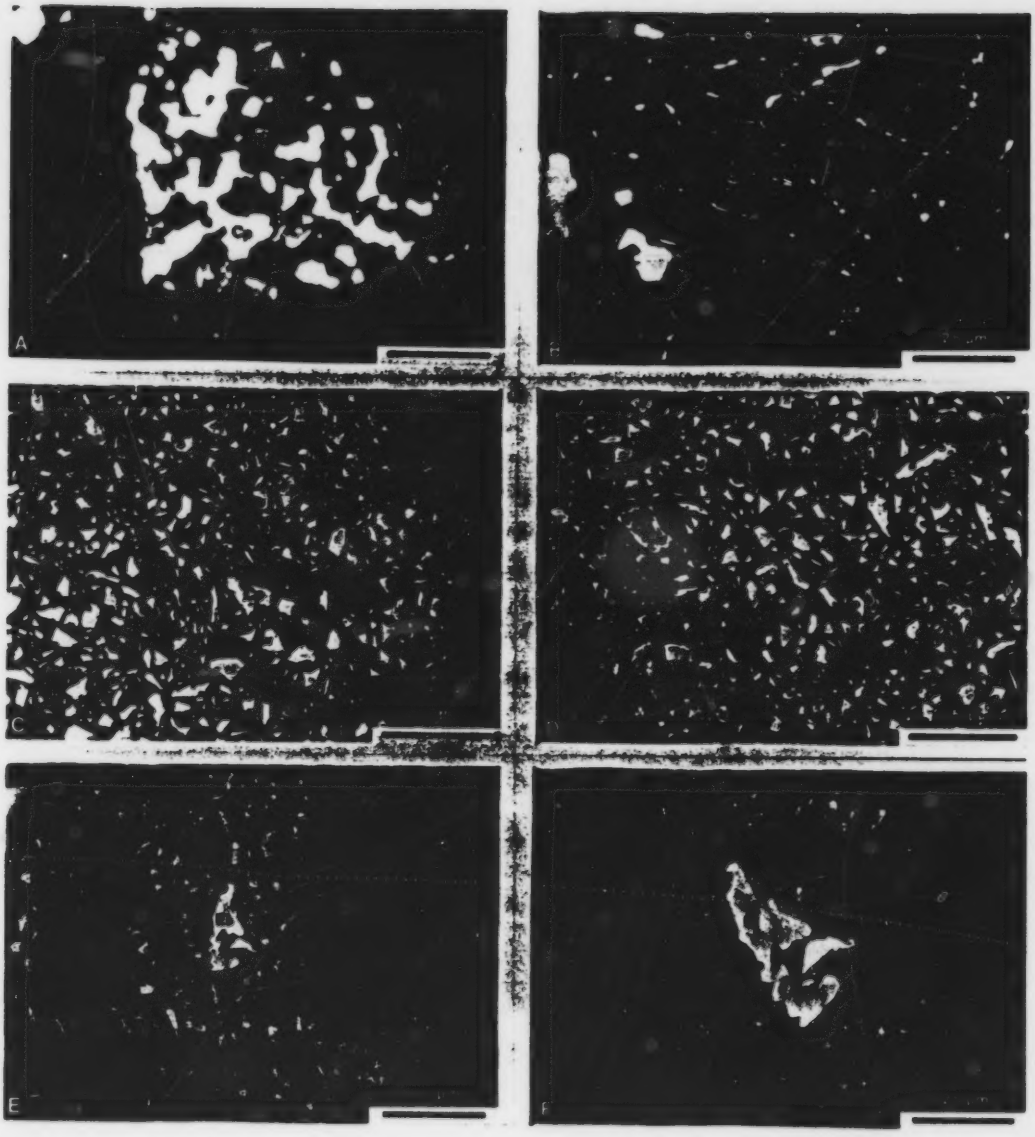
(C) Cu-CONCENTRATE : Abundant free sphalerite grains, and sphalerite locked with chalcopyrite. Otherwise concentrate is 'clean' with only minor pyrite. Negligible silicate or carbonate gangue. PPL.

(D) Cu-CONCENTRATE : Abundant free sphalerite grains, and sub-free and middling sphalerite locked with chalcopyrite. Note diablastic chalcopyrite-sphalerite particle (center) and low pyrite content. PPL.

(E) Cu-CONCENTRATE : Relatively coarse-grained, free type 2 mercurian electrum (El) recovered in the Cu-concentrate. Note sphalerite grains are all essentially free, and the median grain size is approximately 15 to 20 um. PPL.

(F) Cu-CONCENTRATE : Detail of (E) showing a grain of coarse-grained, free type 2 mercurian electrum, which appears to have been milled and repeatedly bent during grinding. OI. PPL.

PLATE I.



Chapter VII

MINERAL LIBERATIONS

In this chapter the liberation and classification of chalcopyrite, sphalerite and galena are evaluated. The liberation (% free) is the volume % of each of these 3 minerals of interest that occur as free grains; grains having no common boundary with other minerals, and hence not touching or locked with grains of other minerals. The remainder, the unliberated grains occur in composite particles with one or more other minerals. The classification describes the volume % of each unliberated grain of the mineral of interest in terms of the total volume of the particle in which that unliberated grain occurs. Thus, the classification describes the composite particles in which the mineral of interest occurs in terms of the unliberated grains contribution to the total volume of the particle. Additionally, several classifications are produced for each mineral of interest in each product. These additional classifications describe the particles containing the mineral of interest (e.g. chalcopyrite) and a secondary mineral of interest (e.g. sphalerite) in terms of the volume % of chalcopyrite in the detected particle. Thus, various categories of particles containing the mineral of interest are selected based on the accompanying mineral(s), and each category is classified. However, the main thrust of this chapter will focus on the principal category of unliberated particles of the minerals of interest e.g. 'chalcopyrite Vs all', where 'all' includes all other minerals, and thus includes all particles in which chalcopyrite occurs. It should be stressed that with the exception of in the Cu-concentrate, neither the grain size distribution nor the liberation of pyrite were determined.

FREE GRAINS

The raw liberation (% free) for chalcopyrite, sphalerite and galena in each of the 25 mill product samples is presented in Table 1 of Appendix B, and are condensed for the 17 principal mill products in Table 19.

The condensed raw liberations were transformed into a coherent set of data by material balance calculations. The liberations were first weighted in terms of the amount of the mineral(s) present in each product. The Cu, Zn and Pb assays were taken as being a more precise equivalent of the modal amounts of chalcopyrite, sphalerite and galena, respectively. Consequently, for each product the Cu assay was multiplied by the liberation for chalcopyrite to produce the modal fraction of the Cu assay occurring as free chalcopyrite. The same was done for Zn and sphalerite, and Pb and galena. These values were transformed by MATBAL into a set of adjusted modal fractions of the respective free mineral, and recoveries for the same were calculated. The adjusted modal fractions are subsequently divided by the respective assay for the

TABLE 19

CONDENSED RAW MINERAL LIBERATIONS IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE	GALENA
1. Feed.	56.4	41.8	7.34
2. Cu-Scav Co.	18.4	14.4	8.57
3. Cu-Roug Co.	60.4	39.7	20.8
4. Cu-Cl.1 Fd.	64.1	34.0	31.0
5. Cu-Cl.1 Co.	65.7	44.9	19.4
6. Cu.Cl.2 Co.	75.4	49.7	18.3
7. Zn-Roug Fd.	3.40	57.4	5.45
8. Zn-Roug Co.	15.6	58.7	27.5
9. Zn-Cl.1 Co.	13.5	61.8	32.3
10. Zn-Cl.2 Co.	0.30	63.2	12.4
11. Cu-Roug Tl.	27.0	43.0	5.70
12. Cu-Scav Tl.	21.9	50.4	6.10
13. Cu-Cl.1 Tl.	43.8	36.6	11.6
14. Cu-Cl.2 Tl.	68.7	65.6	9.63
15. Zn-Cl.1 Tl.	11.5	21.7	34.8
16. Zn-Cl.2 Tl.	7.30	12.3	37.6
17. Zn-Roug Tl.	3.90	0.00	4.92

TABLE 20

MINERAL LIBERATIONS IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE	GALENA
1. Feed.	60.6	50.5	7.69
2. Cu-Scav Co.	21.5	14.4	8.93
3. Cu-Roug Co.	63.2	37.6	17.1
4. Cu-Cl.1 Fd.	63.5	40.2	16.5
5. Cu-Cl.1 Co.	65.8	47.2	19.6
6. Cu.Cl.2 Co.	65.7	42.8	21.9
7. Zn-Roug Fd.	5.83	50.9	7.90
8. Zn-Roug Co.	2.63	54.9	10.5
9. Zn-Cl.1 Co.	3.80	53.1	25.1
10. Zn-Cl.2 Co.	0.63	55.6	7.04
11. Cu-Roug Tl.	12.9	48.3	5.66
12. Cu-Scav Tl.	5.30	51.5	5.67
13. Cu-Cl.1 Tl.	43.7	33.9	13.8
14. Cu-Cl.2 Tl.	68.7	62.2	10.0
15. Zn-Cl.1 Tl.	11.8	21.7	20.9
16. Zn-Cl.2 Tl.	8.45	12.3	42.6
17. Zn-Roug Tl.	8.62	0.05	3.91

NOTE : Calculated above values by dividing the material balance adjusted Cu assays (ideally equivalent to modal amounts of chalcopyrite) by the adjusted modal fractions of the Cu assay as free chalcopyrite (i.e. Cu assay x liberation), both of which datasets were calculated independently by MATBAL.

TABLE 21

RECOVERY OF FREE MINERALS IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE	GALENA
1. Feed.	100.0	100.0	100.0
2. Cu-Scav Co.	2.65	2.36	7.30
3. Cu-Roug Co.	107.5	19.9	71.1
4. Cu-Cl.1 Fd.	112.9	23.8	75.2
5. Cu-Cl.1 Co.	104.6	13.2	41.2
6. Cu.Cl.2 Co.	99.3	9.30	37.2
7. Zn-Roug Fd.	0.88	91.4	103.0
8. Zn-Roug Co.	0.18	91.4	81.3
9. Zn-Cl.1 Co.	0.37	91.9	297.6
10. Zn-Cl.2 Co.	0.04	90.7	41.2
11. Cu-Roug Tl.	3.38	93.1	70.1
12. Cu-Scav Tl.	0.73	90.7	62.9
13. Cu-Cl.1 Tl.	8.27	10.6	34.0
14. Cu-Cl.2 Tl.	5.33	3.91	4.12
15. Zn-Cl.1 Tl.	0.15	0.73	40.1
16. Zn-Cl.2 Tl.	0.33	1.24	256.4
17. Zn-Roug Tl.	0.70	0.01	21.7

NOTE : Calculated by MATBAL as the recovery of the modal fraction of the Cu assay as free chalcopyrite (i.e. Cu assay x liberation).

corresponding metal and product to give adjusted, but non-coherent liberations. Thus, a set of adjusted liberations is not generated, but is extracted from two sets of coherent data, for example, by dividing the adjusted Cu assay by the adjusted modal fraction of the Cu assay as free chalcopyrite. These extracted liberation data are presented in Table 20. It should be stressed again that MATBAL balances each set of variables independently, such that the adjusted data may not necessarily cross-balance. However, ideally the closer the true values are approached for each variable, the closer the cross-balance achieved.

Comparison of the values in Tables 19 and 20, indicate that the derived adjustments to the raw liberations are relatively small as a proportion of the raw values, particularly for chalcopyrite and sphalerite. This implies that the image analysis generated relatively accurate modal fractions of free mineral grains in each sample, or that stereological biases that create systematic errors are expressed relatively consistently. The most significant stereological bias results from the apparent increase in the liberation or % free grains of the mineral of interest, due to the sectioning effect i.e. unliberated grains appear free in the plane of the polished section. Hill et al. (1986) mathematically simulated a 'worst-case' model of this phenomena, and determined that the apparent liberation of the unliberated grains of the mineral of interest, as determined by an area measuring image analyser (e.g. MP-SEM-IPS-IAS), was for example 13.20 % for a grain comprising 70 % of the particle. This effect is lessened in natural systems, where the vast majority of observations fall within two classes of the actual class of the particle. However, all assemblages of particles containing unliberated grains of the mineral of interest will yield some free grains of that mineral, whereas free grains of the same will never appear locked.

Petruk (1978) has observed that between 10 to 15 % of the unliberated grains of the mineral of interest appear as free grains in mill product samples. Petruk (1978) also suggested that the liberation data should be corrected for unliberated grains that appear free. However, it has been found that uncorrected data will yield material balances much more readily than corrected data, and the practise of data correction has been discontinued (W. Petruk, pers. comm. 1987). There is apparently a significant discrepancy between the theoretical and practical considerations of liberation analysis. Consequently, a small (< 15 %) uncorrected portion of the unliberated grains of the mineral(s) of interest appear as free grains, and are incorporated into the material balance adjustments of the liberation data of each sample. The resulting adjustments will reflect the incorporation of this systematic bias. Consequently, the 60.6 %, 50.5 % and 7.69 % free grains of chalcopyrite, sphalerite and galena, respectively include a small, undetermined component of unliberated grains of the respective minerals. As the proportion of actual unliberated grains of the mineral of interest decreases in a product, the smaller the amount of such grains that are available to appear as free grains, and hence the smaller the bias produced. Thus, the adjusted modal fraction, and recoveries of these, become more precise and accurate the higher the actual or real liberation. This and other related effects are observed for the present datasets, and an appreciation of their origins is critical to the interpretation of the data. (See Petruk 1978, and Hill et al. 1986).

The calculated liberations for chalcopyrite, sphalerite and galena in the circuit feed are 60.6 %, 50.5 % and 7.69 %, respectively. These compare with the predicted liberations of 62 % for chalcopyrite, 66 % for sphalerite and 45 % for galena[6] (Healy & Petruk 1988). These latter values are based on average grain size distributions of each of the respective minerals in a representative suite of unbroken ore samples, and a nominal grind of 65 % - 43 um. Thus, the predicted liberations are dependent on several variables: (1) the repesentivity of the ore suite; (2) the actual grind used; and (3) the validity of both preferential and random breakage as the principal mechanisms of breakage during grinding as incorporated into Petruk's Liberation Model (Petruk 1976). The grinding curve for the present Trout Lake circuit feed (sampled on Feb 5th 1986) is defined by 67.8 % -43 um (See SIZE ANALYSIS), and thus approximates to the grind used in the determination of the predicted liberations by Healy & Petruk (1988). The head grades indicated a Cu-rich and Zn- and Pb-poor feed relative to the average Trout Lake feed. Both material balance calculated ore type components of the feed, and hand-sorting of the rod mill feed indicated higher than average Chalcopyrite Stringer ore type and lower Massive Sphalerite and Banded Pyrite + Sphalerite ore type components in the feed (See ORE COMPOSITION OF THE FEED). Thus, the ore suite from which the predicted liberations were generated is not truely representative of the feed examined. Consequently, the higher Chalcopyrite Stringer ore type component, which contains coarse-grained chalcopyrite and relatively fine-grained sphalerite, and the lower Massive Sphalerite and Banded Pyrite + Sphalerite ore type components, which both contain coarse-grained sphalerite and relatively fine-grained chalcopyrite, will disproportionately increase the liberation of chalcopyrite, and lower that of sphalerite relative to that predicted.

Petruk's liberation model assumes that when the ore is ground to the size distribution of a mineral, 50 % of that mineral will be liberated. This model assumes that random breakage and preferential breakage along grain boundaries are equally significant, and that each of the constituent minerals is of equal hardness and grindability. Grain boundaries are obvious focal points of failure during grinding. Liberation models that do not incorporate textural criteria, such as the configuration of grain boundaries, in addition to grain size criteria are thus subject to large uncertainties. In addition, Petruk et al. (1986) observed that soft brittle minerals are concentrated in the finer-sieve fractions of the grind. Thus, soft minerals such as chalcopyrite and sphalerite should achieve higher liberations than predicted by a model that assumes all minerals will grind and be liberated in accordance with the grain size of the respective mineral in the unbroken ore. Furthermore, Smith et al. (1987) observed that the textures and grain size distribution of pyrite in the unbroken ore have a strong control on the size distribution of the resulting float feed. The grain size distribution of pyrite was determined to be finer than that of chalcopyrite and sphalerite in the Trout Lake ore (Healy & Petruk 1988). However, chalcopyrite and sphalerite are concentrated in the finer-grained fractions of the grind, while that of pyrite lies slightly coarser than that of the grind (See SIZE ANALYSIS).

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[6] The predicted liberation for galena was erroneously given by Healy & Petruk (1988) as 52 %. The correct value is in fact 45 %.

Therefore, the prediction of liberation based on grain size distributions of minerals in the unbroken ore, and the grind at which this ore will be processed (Petruk 1976), is based on an idealised or simplified model of an extremely complex mineralogical system. Nonetheless, the model is useful for comparison of essentially similar ores (e.g. Trout Lake, Centennial, South Main and Callinan) where predicted liberations for ores from presently undeveloped mines can be calibrated to actual liberations achieved during processing of already studied ores (e.g. Trout Lake).

The degree and style of intergrowth of the minerals is also critical in determining the degree of liberation, and the nature of the unliberated grains. Because chalcopyrite commonly occurs as fine-grained inclusions in sphalerite, whereas the reverse is much less frequent, chalcopyrite is expected to achieve higher liberation than sphalerite. Particles containing unliberated grains of sphalerite are however likely to contain a higher proportion of fine-grained and volumetrically subordinate grains of chalcopyrite, than sphalerite in particles dominated by chalcopyrite. Consequently, sphalerite does exhibit a lower proportion of absolutely free grains than chalcopyrite, but the unliberated grains of sphalerite comprise a greater proportion of the particles in which they occur (i.e. largely sub-free) than unliberated grains of chalcopyrite (See UNLIBERATED GRAINS). Thus, while chalcopyrite exhibits higher liberation than sphalerite (i.e. 60.6 % versus 50.5 % free grains), the recovery of sphalerite should be equal or greater than that of chalcopyrite based on the expected proportion of sub-free grains (i.e. 91.5 % versus 90.4 % grains in the recoverable classes).

Because the observed liberation of galena in the feed is only 7.69 %, the material balance modelling of free galena is potentially strongly influenced by the component of apparently free grains derived from unliberated grains. Thus, MATBAL may have modelled apparent free grains superimposed on, and masking the actually free grains of galena, resulting in the apparently non-specific behaviour of free galena. For several samples where the liberation is greater than 20 to 30 %, and thus much greater than the amount likely to be derived from actually unliberated galena, it can be assumed that much of the observed free galena is actually free. It might then be assumed from the adjusted liberations for free galena, that free galena concentrates in the Cu-2nd cleaner concentrate, the Zn-1st cleaner concentrate, and the Zn-1st and -2nd cleaner tails. However, such erratic behaviour as concentrating in both the Zn-1st cleaner concentrate and tails, seems unlikely.

On the basis of the above discussion relating to the simulated behaviour of apparently free grains of galena, it is possible to interpret that free galena reports largely to the Cu-concentrate, and actually exceeds the recovery of 37.2 % of the apparently free galena. The high recoveries of apparently free galena in the Zn-circuit (e.g. 256.4 % in the Zn-2nd cleaner tails) reflect apparent free grains stereologically derived from the high circulating loads of sphalerite-galena particles, as well as actual free grains that failed to float initially in the Cu-circuit. The recovery of 21.7 % of the free grains of galena in the tailings is based on only 3.91 % of the galena in the tailings appearing as free grains. A value of 3.91 % (or 4.92 % in the raw data) is readily stereologically derived from an assemblage of unliberated grains. Thus, an interpretation that free galena is not lost to the tailings, but reports principally to the Cu-concentrate is potentially argueable.

The highest % free chalcopyrite was observed in the Cu-2nd cleaner tails, and reflects the small circulating load of free chalcopyrite grains, as well as apparent free grains derived from unliberated grains of chalcopyrite. The latter grains would most probably correspond to sub-free middling particles, which are likely to be partly rejected in the Cu-cleaners, and also yield relatively high percentages of apparently free grains. Nonetheless, and despite apparent liberations of 8.62 % free chalcopyrite in the tails, these values of liberation are small in terms of recovery of free chalcopyrite grains (i.e. 5.33 % and 0.70 % recovery of free chalcopyrite). This is also true of the 0.63 % free chalcopyrite in the Zn-concentrate. In consequence, the recovery of free chalcopyrite in the Cu-concentrate is determined to be a minimum of 99.3 %, which confirms the excellent recovery of free chalcopyrite.

The adjusted liberation data for sphalerite contains few values that are likely to be principally generated by stereological derivation from unliberated grains (i.e. 14.4 % and 12.3 % free sphalerite in the Cu-scavenger concentrate and the Zn-2nd cleaner tails, respectively). Both these values, that may contain biased values of % free sphalerite, correspond to 2.36 % and 1.24 % of the free sphalerite, and are therefore relatively insignificant. The Cu-scavenger tails, which is the Zn-circuit feed recovers 90.7 % of the free sphalerite, virtually all (actually 100 % due to 'rounding-off') of which reports to the Zn-2nd cleaner concentrate (i.e. 90.7 %). Thus, essentially all of the free sphalerite that enters the Zn-circuit is recovered. The Cu-2nd cleaner concentrate contains 42.8 % free sphalerite and corresponds to the remaining 9.30 % of lost free sphalerite. Values of % free sphalerite such as 47.2 % and 42.8 % in the Cu-1st and 2nd cleaner concentrates, respectively, cannot be generated by stereological biases, even in the 'worst case' simulations of Hill et al. (1986). Even while accepting that a large component of this % free sphalerite may be derived from observations of unliberated grains, these grains would necessarily be in sub-free classes, in order to appear free in such amounts, and thus correspond to recoverable classes of locked sphalerite, regardless. Thus, the liberation data for sphalerite demands the interpretation that approximately 9.3 % of the free sphalerite grains are lost to the Cu-concentrate, while the remainder reports almost exclusively to the Zn-concentrate.

Flotation efficiencies (as defined in MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES) were calculated for free chalcopyrite, sphalerite and galena, and are given in Table 22. These flotation efficiencies are presented as curves for the Cu- and Zn-circuit units in Figure 7. The flotation efficiencies of chalcopyrite and sphalerite free grains are interpreted to reflect real flotation behaviour, whereas that of free galena is probably strongly biased by stereological derivation of apparent free grains.

TABLE 22

FLOTATION EFFICIENCIES FOR FREE CHALCOPYRITE, SPHALERITE AND GALENA.

PRODUCT.	Chalcopyrite Recovery			Sphalerite Recovery			Galena Recovery		
	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.
Cu Scavenger.	3.38	2.65	78.4	93.1	2.36	2.54	70.1	7.30	10.4
Cu Rougher.	110.9	107.5	96.9	113.0	19.9	17.6	141.3	71.1	50.3
Cu 1st Cleaner.	112.9	104.6	92.7	23.8	13.2	55.5	75.2	41.2	54.8
Cu 2nd Cleaner.	104.6	99.32	94.9	13.2	9.30	70.5	41.2	37.25	90.3
Zn Rougher.	0.88	0.18	20.5	91.42	91.43	100.0	103.0	81.3	78.9
Zn 1st Cleaner.	0.51	0.37	72.5	92.6	91.9	99.2	337.7	297.6	88.1
Zn 2nd Cleaner.	0.37	0.04	10.8	91.9	90.7	98.7	297.6	41.2	13.8

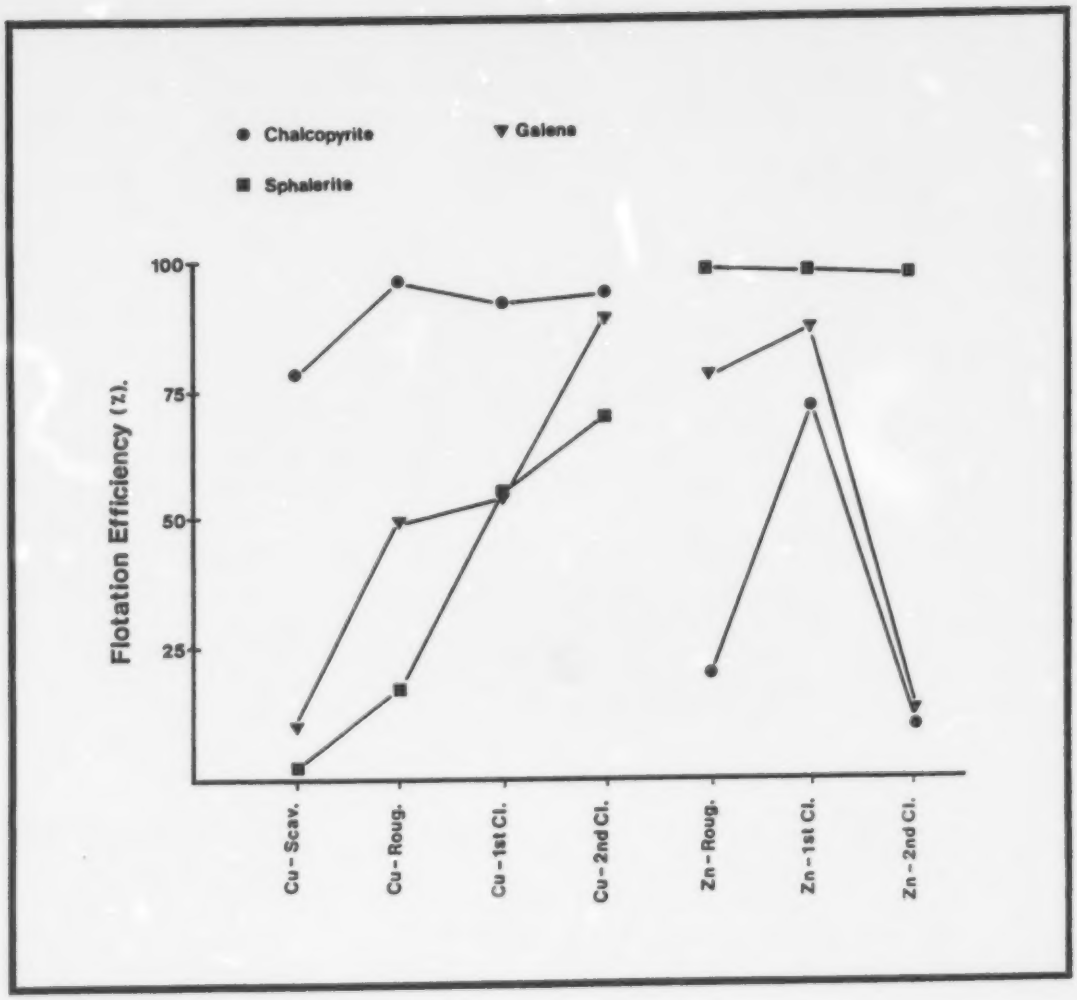


FIGURE 7.
PLOT OF FLOTATION EFFICIENCIES FOR FREE CHALCOPYRITE, SPHALERITE AND GALENA.

CHALCOPYRITE

With the exception of the Cu-scavenger, all the units of the Cu-circuit exhibit chalcopyrite flotation efficiencies of greater than 92 %. It is improbable that the Cu-scavenger, which by definition is 'pulling hard' would not recover a greater proportion of the free chalcopyrite presented to it as the Cu-2nd cleaner, which will in fact reject some free chalcopyrite in the process of rejecting undesirable contaminants. Because the Cu-scavenger is principally recovering middling particles and sub-free particles of chalcopyrite, which failed to float initially, significant observations of apparent free grains are likely from this assemblage of unliberated chalcopyrite. In addition, because inevitably observations of apparent free grains occur in products from the Zn-circuit, the chalcopyrite liberation in the Cu-scavenger tails (the Zn-circuit feed) is adjusted to account for these apparent losses. Thus, this lower flotation efficiency for free chalcopyrite in the Cu-scavenger, is at least partly based on the proportionally higher amount of apparent free grains in products containing negligible actual free chalcopyrite, but which account for only a minute modal fraction of the free chalcopyrite.

The flotation efficiencies of free chalcopyrite for the Zn-circuit units, are based on values of adjusted % free chalcopyrite that all fall below 12 %, and are therefore potentially subject to biases by observations of apparent free grains derived from the dominant assemblage of unliberated grains in the constituent products. The recovery of free grains of chalcopyrite in each product of the Zn-circuit is less than 1 %, and confirms the insignificance of these biased observations. Therefore, the behaviour of apparent free grains of chalcopyrite in the Zn-circuit is interpreted as being largely of an extraneous nature, except that it may partly reflect coarse-grained, sub-free grains of chalcopyrite that partly appear as free grains, and actual over-sized, free chalcopyrite. Accepting this cautiously, this type of chalcopyrite is principally lost at the Zn-rougher stage, and is virtually completely rejected by the Zn-2nd cleaner.

SPHALERITE

The flotation efficiencies for free sphalerite through the Cu-circuit steadily increase to 70.5 % in the Cu-2nd cleaner. With the exception of the Cu-scavenger, whose concentrate recovers only 2.36 % of the apparent free sphalerite in the feed, these values are interpreted to be essentially bias-free. This confirms that a fraction of the free sphalerite in the feed is recovered through the Cu-circuit, with progressively lower rates of rejection. The initial recovery of this free sphalerite does not occur at the Cu-scavenger, but at the Cu-rougher stage, implying that this free sphalerite fraction is not characterised by marginal flotability, but is readily floated.

The flotation efficiencies for free sphalerite through the Zn-circuit drop from essentially 100 % to 98.7 % at the Zn-2nd cleaner. Thus, the recovery of free sphalerite grains entering the Zn-circuit is virtually total, even without correcting for the depression of the values by the inevitable observations of at least a few apparent free grains in the tails of these

units. Therefore, the activation of sphalerite by $CuSO_4$, and the flotation using the Minerec collector is extremely effective in the recovery of free sphalerite.

GALENA

The flotation efficiencies for apparent free galena, though potentially strongly biased by stereological derivation from unliberated grains, permit certain tentative conclusions. The progressive increase in flotation efficiencies through the Cu-circuit from 10.4 % to 8[illegible]3 % in the Cu-2nd cleaner may reflect the consistent recovery of free galena, and the progressive rejection of the initially dominant fraction of apparent free grains stereologically derived from unliberated grains. By corollary, the flotation efficiencies for actually free galena in the Cu-circuit are probably all greater than the 75 % percentile, but are depressed by abundant observations of apparently free grains in the corresponding tails of each unit. It may be interpreted therefore that actually free galena is largely recovered in the Cu-concentrate, though free galena represents less than 8 % of the galena in the feed. The simulated behaviour of apparently free galena in the Zn-circuit may be interpreted to be extraneous, in that it may represent largely apparent free grains derived from unliberated grains. The loss of 21.7 % of the apparently free galena to the tailings can be attributed to these biased observations, and it is possible that free galena is not lost to the tailings in such significant amounts.

UNLIBERATED GRAINS

In this section, the modal fraction of the minerals of interest that occur as grains locked with other minerals (i.e. the % unliberated), and the classification of these unliberated grains are evaluated. The classifications and grain size distributions of chalcopyrite, sphalerite and galena occurring in selected categories of particles that contain all other minerals, or just specific accompanying mineral(s) were determined for each product. The grain size data is treated more completely in the next chapter (See SIZE ANALYSIS), though it will be touched upon in this section. Nonetheless, the analyses presented in this section constitute the most extensive and comprehensive aspect of the study. The related raw data tables and plots comprise Part C and D (i.e. pages 14 to 351) of Volume 2, the appendix of raw image analysis data.

It has been shown that the systematic stereological bias that yields observations of apparent free grains derived from actually unliberated grains of the mineral of interest, generates slightly elevated liberations of that mineral (See FREE GRAINS). By corollary the observed % unliberated is correspondingly depressed. Hill et al. (1986) note that this bias is minimised in assemblages consisting of complex particles, i.e. many mineral grains in each particle, and non-planar intergranular boundaries. Petruk (1978) determined that less than 15 % of the unliberated grains yield observations of free grains, while Hill et al. (1986) point out that the

window between corrected and uncorrected (i.e. actual and apparent) is approximately 10 % or less for a nominal particle assemblage. However, while the % unliberated is depressed, the compositional integrity or classification of the observed unliberated grains is essentially preserved. This preservation of the integrity of the actual classification results from the amount of grains in each class that appear in adjacent classes, being at least partially offset by the equivalent amount appearing in that class, but originating in adjacent classes (Hill et al. 1986). Thus, while a one-way stereological bias produces depressed values for the % unliberated of a mineral, the classification of the observed unliberated grains is assumed to be true.

The bulk of the following sections on the liberation and classification of chalcopyyrite, sphalerite and galena is based on the classification of each of these minerals in particles containing at least one grain of one or more of any of the other detected minerals (e.g. chalcopyrite Vs all), and the material balance calculations on these data. Thus, the modal amount of chalcopyrite occurring in chalcopyrite Vs all particles equals the % unliberated for that product, while the chalcopyrite Vs all classification corresponds to the modal fraction in each class of unliberated chalcopyrite. The material balance calculations were performed on the modal fraction of the mineral of interest that occurs (1) as free grains, (2) as unliberated grains, (3) as unliberated grains in each class of particle, and (4) as unliberated grains in selected composite classes of particles. A set of adjusted modal fractions, and the recovery of each such fraction in each product were thus generated and evaluated. The additional classifications where particles containing the mineral of interest (e.g. chalcopyrite) and a secondary mineral of interest (e.g. chalcopyrite Vs gangue) are discussed at the end of each section on the respective mineral of interest. Material balance calculations were not performed on these additional classifications, and so the discussion deals with the raw modal fraction of the mineral of interest occurring in particles that also contain these secondary minerals of interest e.g. the amount of unliberated sphalerite that is locked in composite particles with chalcopyrite in the Cu-concentrate.

The % unliberated of chalcopyrite corrseponds ideally to the modal fraction of chalcopyrite occurring in chalcopyrite Vs all particles, or the difference between the total amount of chalcopyrite and that of free chalcopyrite. Because these two measures of the % unliberated are determined differently the values are subject to differences in uncertainties and are commonly slightly inequivalent. In order to model the % unliberated, the difference between the modal amount of the mineral of interest and that of free grains of the same mineral were used as the raw data. Thus, they are the inverse of the raw % free data of the same mineral. The raw % unliberated data for chalcopyrite, sphalerite and galena in each of the 25 mill product samples is presented in Table 2 of appendix B, and are condensed for the 17 mill products in Table 23. As was done with the % free data, the different modal fractions (e.g. the total % unliberated chalcopyrite or the % unliberated in the 70-80 % class) of the mineral of interest were first weighted in terms of the amount of the respective mineral present in each product. The Cu, Zn and Pb assays were taken as being a more precise equivalent of the modal amount of chalcopyrite, sphalerite and galena, respectively. Thus, for each product the Cu assay was multiplied by the modal fraction of chalcopyrite occurring as grains in all, in each class, or in composite classes of unliberated chalcopyrite. The same

was done for Zn and sphalerite, and Pb and galena. This 'mineral-metal' data (i.e. modal fraction of mineral x metal assay) was transformed by MATBAL into a coherent set of adjusted modal fractions of the respective metal, and recoveries of the same were calculated. The calculated % unliberated chalcopyrite, sphalerite and galena were extracted from the adjusted modal fraction of the respective metal assay by dividing by the adjusted metal assay, and these are presented in Table 24. Thus, the adjusted values for % free and % unliberated chalcopyrite are based on (1) the adjusted Cu assay, (2) the adjusted modal fraction of the Cu assay as free chalcopyrite, and (3) the adjusted modal fraction of the Cu assay as unliberated chalcopyrite; three independently material balanced variables. In Table 3 of appendix B the calculated % free and % unliberated were combined for each of the three minerals in each of the products, so as to assess the degree of convergence of the adjustments i.e. ideally if the adjustments approach the true values, the sum should equal 100. Clearly a value of 104.3 % unliberated chalcopyrite in the tailings is slightly elevated, and reflects the slight deviation of each of the adjusted variables from the true value. However, from this table it is apparent that the convergence is generally acceptable (i.e. ± 5 % of 100), particularly for products containing abundant modal amounts of the mineral(s), and thus benefitting from improved counting statistics i.e. 98.2 % sphalerite in the Cu-2nd cleaner concentrate.

The recoveries of the % unliberated chalcopyrite, sphalerite and galena, determined as the recovery of the modal fraction of the metal assay occurring as unliberated grains of the respective mineral, are presented in Table 25. While approximately 40 % of the chalcopyrite in the feed is unliberated, 79.7 % of this is recovered in the Cu-concentrate, with 8.53 % reporting to the Zn-concentrate and 11.8 % to the tailings. Thus, while 99.3 % of the free grains are recovered, and additional 79.7 % of the unliberated grains are also recovered in the Cu-concentrate. The remaining unliberated grains are split approximately 40 : 60 between the Zn-concentrate (as chalcopyrite-disease and as granular and diablastic chalcopyrite-sphalerite particles) and the tailings (principally as pyrite-chalcopyrite and gangue-chalcopyrite particles).

While approximately 50 % (i.e. raw = 58.2 % and adjusted = 46.1 %) of the sphalerite in the feed is unliberated, 72 % of this sphalerite is recovered in the Zn-concentrate, with 14.8 % lost to the tailings. The remainder of 13.2 % is recovered in the Cu-concentrate, and is slightly greater than the recovery of 9.30 % of the free sphalerite in the Cu-concentrate. In as much as the 9.30 % recovery of free sphalerite in the Cu-concentrate is anomalous, the somewhat higher recoveries of unliberated sphalerite may be partly attributable to composite particles of chalcopyrite and sphalerite. In this regard, it is noteworthy that while 2.36 % of the free sphalerite is recovered in the Cu-scavenger concentrate, 15.8 % of the unliberated sphalerite is recovered in the Cu-scavenger concentrate, and is interpreted to reflect the recovery of these middling particles of chalcopyrite and sphalerite. The recoveries/losses of 11.8 % and 14.8 % of unliberated chalcopyrite and sphalerite, respectively to the tailings, represents greater than 90 % and 95 % of the chalcopyrite and sphalerite, respectively in the tailings (i.e. only 0.70 % and 0.01 % of the observed free chalcopyrite and sphalerite, respectively are lost to the tailings).

TABLE 23

CONDENSED RAW % UNLIBERATED MINERAL IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE	GALENA
1. Feed.	43.6	58.2	92.7
2. Cu-Scav Co.	81.6	85.6	91.4
3. Cu-Roug Co.	39.6	60.3	79.2
4. Cu-Cl.1 Fd.	35.9	66.0	69.0
5. Cu-Cl.1 Co.	34.3	55.1	80.6
6. Cu.Cl.2 Co.	24.6	50.3	81.7
7. Zn-Roug Fd.	96.6	42.6	94.5
8. Zn-Roug Co.	84.4	41.3	72.6
9. Zn-Cl.1 Co.	86.5	38.2	67.7
10. Zn-Cl.2 Co.	99.7	36.7	87.6
11. Cu-Roug Tl.	73.0	57.0	94.3
12. Cu-Scav Tl.	78.1	49.6	93.9
13. Cu-Cl.1 Tl.	56.2	63.4	88.4
14. Cu-Cl.2 Tl.	31.3	34.4	90.4
15. Zn-Cl.1 Tl.	88.5	78.3	65.2
16. Zn-Cl.2 Tl.	92.7	87.7	62.4
17. Zn-Roug Tl.	96.1	100.0	95.1

TABLE 24

ADJUSTED % UNLIBERATED MINERAL IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE	GALENA
1. Feed.	35.8	46.1	90.2
2. Cu-Scav Co.	75.9	87.4	92.0
3. Cu-Roug Co.	34.1	61.6	78.6
4. Cu-Cl.1 Fd.	34.0	58.8	79.6
5. Cu-Cl.1 Co.	31.1	50.8	79.6
6. Cu.Cl.2 Co.	31.1	55.4	77.4
7. Zn-Roug Fd.	86.9	45.6	87.9
8. Zn-Roug Co.	88.1	41.1	78.7
9. Zn-Cl.1 Co.	89.3	43.0	71.8
10. Zn-Cl.2 Co.	88.0	40.3	83.4
11. Cu-Roug Tl.	81.7	48.6	91.6
12. Cu-Scav Tl.	86.8	45.0	91.8
13. Cu-Cl.1 Tl.	57.9	66.0	79.6
14. Cu-Cl.2 Tl.	31.4	35.0	88.9
15. Zn-Cl.1 Tl.	88.7	78.2	64.5
16. Zn-Cl.2 Tl.	91.4	86.2	60.6
17. Zn-Roug Tl.	85.9	104.3	100.6

NOTE : Calculated above values by dividing the material balance adjusted Cu assays (ideally equivalent to modal amounts of chalcopyrite) by the adjusted modal fractions of the Cu assay as unliberated chalcopyrite (i.e. Cu assay x % unliberated), both of which datasets were calculated independently by MATBAL.

TABLE 25

RECOVERY OF UNLIBERATED MINERALS IN THE 17 MILL PRODUCTS.

PRODUCT.	CHALCOPYRITE	SPHALERITE	GALENA
1. Feed.	100.0	100.0	100.0
2. Cu-Scav Co.	15.8	15.8	6.36
3. Cu-Roug Co.	98.3	35.9	28.0
4. Cu-Cl.1 Fd.	102.4	38.3	31.1
5. Cu-Cl.1 Co.	83.9	15.6	14.4
6. Cu.Cl.2 Co.	79.7	13.2	11.2
7. Zn-Roug Fd.	22.2	89.7	99.4
8. Zn-Roug Co.	10.4	74.9	52.4
9. Zn-Cl.1 Co.	14.6	81.5	73.0
10. Zn-Cl.2 Co.	8.53	72.0	41.8
11. Cu-Roug Tl.	36.1	102.6	95.1
12. Cu-Scav Tl.	20.3	86.8	88.8
13. Cu-Cl.1 Tl.	18.6	22.7	16.8
14. Cu-Cl.2 Tl.	4.13	2.41	3.14
15. Zn-Cl.1 Tl.	1.89	2.88	10.6
16. Zn-Cl.2 Tl.	6.05	9.50	31.2
17. Zn-Roug Tl.	11.8	14.8	47.0

NOTE : Calculated as the recovery of the modal fraction of the Cu assay as unliberated chalcopyrite (i.e. Cu assay x % unliberated).

Unlike the modelled behaviour of free galena grains which is potentially obscured by stereological biases, that of unliberated galena grains is interpreted to be essentially true and representative. In addition, it represents approximately 95 % of the galena in the feed, and is thus the only significant component of these two. Only 11.2 % of the unliberated galena is recovered in the Cu-concentrate, and most probably reflects sub-free galena grains, and galena locked with chalcopyrite, which is expected to be minor owing to the association of galena in sphalerite-rich, chalcopyrite-poor ore types (Healy & Petruk 1988). The remaining 88.8 % of the unliberated galena reports to the Zn-circuit, and is split almost equally between the Zn-concentrate (41.8 %) and the tailings (47.0 %). The unliberated galena reporting to the Zn-concentrate consists principally of sphalerite-galena composite particles, whereas that reporting to the tailings is interpreted to consist principally of pyrite-galena composite particles.

CHALCOPYRITE

The raw liberation and classification data for chalcopyrite in each of the 25 mill product samples (compiled from the raw % free and raw chalcopyrite Vs all classifications) are given in Table 4 of appendix B, and are condensed for the 17 mill products in Table 26. Each of the values in Table 26 were multiplied by the corresponding adjusted Cu assay for that product (See CHEMICAL ANALYSIS) to yield mineral-metal data representing the raw modal fraction of the adjusted Cu assay occurring as (1) unliberated grains; (2) as unliberated grains in each class; and (3) as free grains. This mineral-metal data is given in Table 5 of appendix B, and the data for each of the ten classes of unliberated chalcopyrite formed the input data for processing by MATBAL. The specific purpose of the later was to generate adjusted mineral-metal data values, from which adjusted modal amounts of unliberated chalcopyrite in each class could be extracted, and to determine the recovery of each class of unliberated chalcopyrite in the various products throughout the circuit.

Before proceeding to deal with the results of the material balance calculations of unliberated chalcopyrite, it is worth examining the raw liberation and classification data for chalcopyrite in the feed (See Table 26). While 56.4 % of the chalcopyrite is free another 29.7 % occurs in the 70-99.9 classes (i.e. sub-free), and thus only 13.9 % occurs in the middling and sub-middling classes (See Fig. 8a). Thus, the counting statistics for chalcopyrite in the individual middling and sub-middling classes are poor, not only in the feed, but particularly also in products where these classes of unliberated chalcopyrite have been preferentially depleted. Thus, while material balance calculations on the individual classes of sub-free chalcopyrite are expected to be both precise and accurate, those on the individual middling and sub-middling classes are expected to yield less satisfactory results.

The calculated recovery of the free, of the unliberated, and of each class of the chalcopyrite classification in each product is presented in Table 27. The data in the columns labelled 'Cu Recovery', '% Locked' and '% Free' have been introduced in previous sections and are presented here for completeness

and ease of comparison. Each of the individual classes with the exception of the 0-10, 20-30 and 90-99.9 exhibit negative recoveries in at least one product. This phenomenon is most pronounced in the middling classes, where poor counting statistics result in unsatisfactory material balances. Nonetheless, certain trends are apparent. Large circulating loads of unliberated chalcopyrite in the middling classes (20-30 to 60-70 particularly) exist at the Cu-rougher and Cu-1st cleaner stages of the Cu-circuit. However, as these classes comprise approximately 10 % of the chalcopyrite in the feed, the circulating loads of chalcopyrite and even unliberated chalcopyrite (all classes) are comparitively low. The recovery in the Cu-2nd cleaner concentrate is consistently above the 99 % level in the sub-free classes, with consequently negligible losses to the tailings. The calculated recoveries in the Cu-2nd cleaner concentrate for the 20-30 to 50-60 classes are distorted by the poor counting statistics with resulting unsatisfactory balances, many of which are in excess of 100 %, and are thus in error. The recoveries in the Cu-concentrate of chalcopyrite in the 0-10 and 10-20 classes are 8.77 % and 12.3 %, respectively, and 45.1 % and 88.1 % respectively, in the Zn-concentrate.. This undoubtedly reflects the predominance of persistent chalcopyrite-disease in sphalerite as the principal form of unliberated chalcopyrite in these two classes, and its recovery principally in the Zn-concentrate (See Fig. 8B).

The adjusted modal fractions of the Cu assay occurring in composite classes of unliberated chalcopyrite and free chalcopyrite are presented in Table 6 of appendix B. By combining the individual classes of the classifications into composite classes with relatively uniform and predictable flotation characteristics, the influence of poor counting statistics on individual classes are reduced in the input data, and satisfactory material balances were generally obtained. The calculated recovery for these composite classes are presented in Table 28. It is apparent from this table that only the middling (i.e. 30-70) classes persisted in yielding poor material balances, with for example a '- 44.6 %' recovery in the tailings. The results of the material balance for this composite class are rejected.

It can be seen that 99.3 % of the free and 79.7 % of the unliberated chalcopyrite is recovered in the Cu-concentrate, resulting in a corresponding 91.6 % Cu recovery (See Fig. 8c). The principal losses occur in the sub-middling classes with only 15 % recovered in the Cu-concentrate, yet 47.6 % reporting to the Zn-concentrate (principally as chalcopyrite-disease) and 37.4 % to the tailings (principally as pyrite-chalcopyrite and gangue-chalcopyrite particles. The middling particles failed to yield a satisfactory balance, whereas the combined middling and sub-middling classes did. Of these combined classes of unliberated chalcopyrite, 27.0 % is recovered in the Cu-concentrate, while the remainder is relatively evenly split between the Zn-concentrate (35.5 %) and the tailings (37.5 %). The middling particles containing chalcopyrite that report to the Zn-concentrate consist principally of (1) diablastic sphalerite-chalcopyrite, (2) normal granular sphalerite-chalcopyrite, (3) pyrite-chalcopyrite, and (4) gangue-chalcopyrite. It is noteworthy that while only 27 % of the combined middling and sub-middling classes are recovered in the Cu-concentrate, 51.9 % are recovered in the Cu-1st cleaner tailings, which has a 'stream flow rate' or product weight of 5.67. Thus, less than approximately 5 % (i.e. the combined middling and sub-middling classes only comprise approximately 10 % of the chalcopyrite) of the

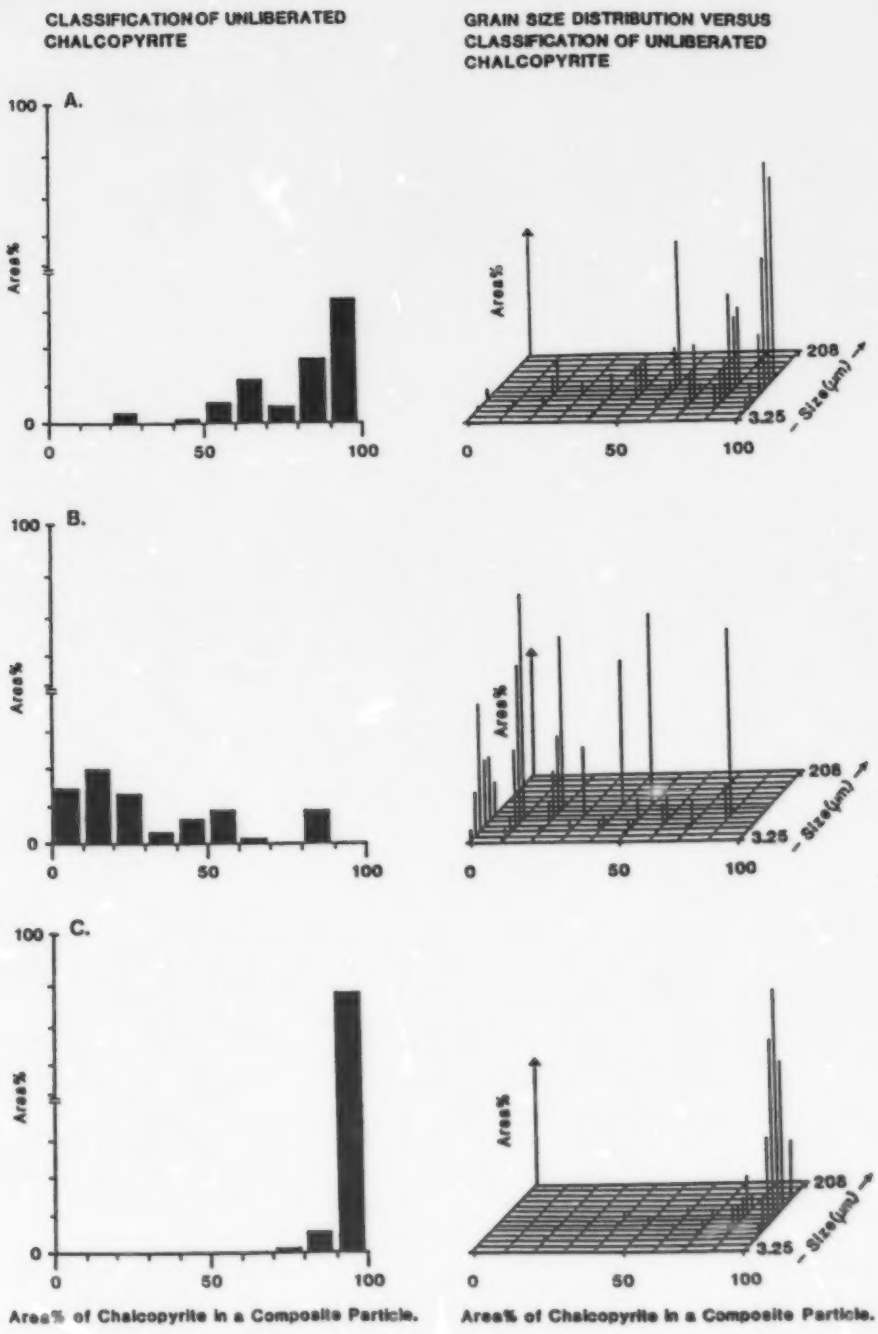


FIGURE 8.
PLOTS OF THE RAW CHALCOPYRITE CLASSIFICATION IN (A) THE FLOAT FEED, (B) THE Zn-CONCENTRATE, AND (C) THE Cu-CONCENTRATE.

TABLE 26

CONDENSED RAW LIBERATION AND CLASSIFICATION DATA FOR CHALCOPYRITE IN THE 17 MILL PRODUCTS.

PRODUCT.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.									% FREE
		0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	43.6	0.84	0.31	1.74	0.71	0.95	3.09	6.10	3.02	9.26	17.4	56.4
2. Cu-Scav Co.	81.6	3.54	4.34	4.07	1.27	2.11	3.18	0.85	2.31	2.04	58.1	18.4
3. Cu-Roug Co.	39.6	0.63	0.46	1.06	0.95	0.93	0.66	1.76	1.70	3.80	27.7	60.4
4. Cu-Cl.1 Fd.	35.9	0.73	0.75	1.16	1.15	1.16	0.28	0.55	0.72	1.41	27.9	64.1
5. Cu-Cl.1 Co.	34.3	0.29	0.65	0.49	0.79	0.21	0.49	0.61	0.77	2.03	27.9	65.7
6. Cu-Cl.2 Co.	24.6	0.10	0.19	0.18	0.14	0.32	0.30	0.42	0.55	1.78	20.7	75.4
7. Zn-Roug Fd.	96.6	57.4	1.85	16.7	1.79	0.00	0.00	0.00	0.00	0.00	19.0	3.40
8. Zn-Roug Co.	84.4	16.0	15.0	7.44	5.87	4.10	4.01	1.49	1.91	2.87	26.0	15.6
9. Zn-Cl.1 Co.	86.5	20.8	8.61	4.54	9.62	1.93	12.9	3.36	0.40	4.40	19.9	13.5
10. Zn-Cl.2 Co.	99.7	18.5	24.5	16.4	4.23	8.73	11.9	2.45	1.56	11.5	0.00	0.30
11. Cu-Roug Tl.	73.0	16.8	14.1	0.28	4.42	33.6	0.25	0.00	0.00	3.48	0.00	27.0
12. Cu-Scav Tl.	78.1	11.4	16.5	1.21	3.19	2.79	8.11	0.48	24.0	0.20	10.4	21.9
13. Cu-Cl.1 Tl.	56.2	4.33	6.38	5.73	3.34	3.00	1.72	3.39	1.67	1.74	26.8	43.8
14. Cu-Cl.2 Tl.	31.3	1.24	1.32	1.32	1.22	1.74	1.91	0.82	1.37	1.11	19.3	68.7
15. Zn-Cl.1 Tl.	88.5	17.0	8.77	5.77	1.74	21.8	1.64	1.55	0.00	0.00	30.3	11.5
16. Zn-Cl.2 Tl.	92.7	9.79	10.4	8.32	1.01	3.70	0.21	1.09	12.8	5.54	39.8	7.30
17. Zn-Roug Tl.	96.1	26.7	2.66	0.00	0.00	25.5	0.00	0.00	0.00	41.5	0.00	3.90

TABLE 27

RECOVERY OF THE FREE, UNLIBERATED AND OF EACH CLASS OF UNLIBERATED CHALCOPYRITE IN THE 17 MILL PRODUCTS.

PRODUCT.	Cu RECOVERY	% LOCKED (0-99.9)	CLASSIFICATION OF UNLIBERATED GRAINS. 0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	% FREE (99.9-100)
1. Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	7.47	15.8	22.4	35.4	0.05	65.5	86.4	6.97	0.41	-0.03	10.4	0.00	2.65
3. Cu-Roug Co.	103.1	98.3	51.9	63.6	456.8	407.2	347.2	172.8	179.2	129.9	111.6	115.0	107.5
4. Cu-Cl.1 Fd.	107.8	102.4	57.1	70.7	488.5	445.5	384.1	207.3	188.4	140.1	114.5	119.3	112.9
5. Cu-Cl.1 Co.	96.3	83.9	13.9	19.3	129.6	143.3	166.3	139.5	108.5	109.4	102.6	104.3	104.6
6. Cu-Cl.2 Co.	91.6	79.7	8.77	12.3	97.9	105.1	129.4	105.1	99.3	99.2	99.8	99.9	99.3
7. Zn-Roug Fd.	9.12	22.2	101.5	94.7	24.7	2.87	2.34	1.92	1.46	0.78	0.25	1.38	0.88
8. Zn-Roug Co.	4.23	10.4	55.4	95.1	22.8	0.51	78.9	0.25	0.62	0.17	7.52	1.36	0.18
9. Zn-Cl.1 Co.	5.84	14.6	64.4	101.8	102.5	6.20	83.9	-2.09	5.52	5.62	14.8	4.88	0.37
10. Zn-Cl.2 Co.	3.4	8.53	45.1	88.1	0.16	-7.41	47.2	-6.74	-0.14	0.16	7.52	0.00	0.04
11. Cu-Roug Tl.	15.8	36.1	113.7	123.1	2.11	60.5	56.9	1.90	1.12	0.74	10.7	0.02	3.38
12. Cu-Scav Tl.	8.36	20.3	91.2	87.7	2.07	-5.05	-29.4	-5.07	0.71	0.76	0.25	0.02	0.73
13. Cu-Cl.1 Tl.	11.5	18.6	43.2	51.3	358.9	302.2	217.8	67.8	79.9	30.7	11.8	15.0	8.27
14. Cu-Cl.2 Tl.	4.70	4.13	5.13	7.01	31.6	38.3	36.9	34.5	9.21	10.2	2.89	4.33	5.33
15. Zn-Cl.1 Tl.	0.76	1.89	10.3	6.97	22.7	7.92	31.7	7.00	0.76	0.01	0.01	1.36	0.15
16. Zn-Cl.2 Tl.	2.37	6.05	19.3	13.7	102.4	13.6	36.7	4.65	5.66	5.46	7.32	4.87	0.33
17. Zn-Roug Tl.	4.89	11.8	46.1	-0.39	1.90	2.36	-76.6	1.67	0.84	0.60	-7.27	0.02	0.70

chalcopyrite is preferentially concentrated as middling and sub-middling particles in the Cu-1st cleaner tailings, and may warrant regrinding.

While as stated earlier, the recovery of free chalcopyrite is 99.3 % in the Cu-concentrate, that of sub-free (i.e. 70-99.9) chalcopyrite is 91.2 % (See Fig. 8c). The principal loss of sub-free chalcopyrite occurs in the tailings (7.21 %), and based on petrographic observation consists principally of pyrite-chalcopyrite and gangue-chalcopyrite particles, with no observations of coarse-grained essentially free (i.e. 90-99.9 class) or free chalcopyrite, as observed in the Zn-concentrate. The combined essentially free class (which includes free grains that may have been detected as being unliberated due to unfiltered artifacts) and the free chalcopyrite, which represents approximately 75 to 80 % of the chalcopyrite, exhibit 99.4 % recovery in the Cu-concentrate. This value may be slightly elevated, in that few of the coarse-grained essentially free chalcopyrite grains in the Zn-concentrate were included in the image analysis of that sample and hence incorporated in the material balance calculations. The combined free and sub-free classes (i.e. 70-100) represent the recoverable categories of chalcopyrite, and of which 97.3 % is recovered in the Cu-concentrate. Despite the possibility that the above values may be slightly elevated due to the inadvertant failure to include the coarse-grained essentially free chalcopyrite in the Zn-concentrate in the image analysis of that product, the recovery of these combined classes is nonetheless excellent. It is clear that in order to improve Cu recovery at acceptable concentrate grades, (1) the coarse-grained essentially free chalcopyrite that reports to the Zn-concentrate must be addressed, and (2) regrinding of the middling (and sub-middling) classes of chalcopyrite must be examined. Regrinding of the Cu-1st cleaner tailings, in which the latter and possibly the former composite classes preferentially concentrate, may yield improved Cu-recoveries. However, as the middling classes of chalcopyrite comprise approximately 5 % or less of the chalcopyrite, and as approximately 50 % (estimated in the absence of a satisfactory material balance) of the middling classes of chalcopyrite are recovered presently, the improvement in the Cu-recovery is expected to be minimal, and considerably less than approximately 2 %, pending on the grind achieved.

The recovery of the adjusted Cu assay in terms of the adjusted modal fraction of chalcopyrite as free grains, unliberated grains and in composite classes of unliberated grains in each product is presented in Table 29. Because many of the individual classes of unliberated chalcopyrite yielded poor material balances (one or more negative values), and because adjusted classifications are not automatically generated, but must be manually extracted from the adjusted mineral-metal data, values for composite classes only were generated. This was done by multiplying the Cu recovery in a product by the adjusted modal fraction of chalcopyrite occurring in the specific class of unliberated and/or free chalcopyrite. Thus, the value of 87.90 % under '70-100' grains in the Cu-2nd cleaner concentrate implies that 87.90 out of the 91.6 % Cu recovery (i.e. 96 %) is attributable to free and sub-free or 'recoverable' chalcopyrite. Similarly, the value of 1.94 % under '70-100' grains in the tailings implies that 1.94 % of the Cu in the feed reports to the tailings as free and sub-free chalcopyrite.

Each one of the composite classes of unliberated and/or free chalcopyrite (e.g. 70-100), yielded good material balances, except the middling classes

TABLE 28

RECOVERY OF THE FREE, UNLIBERATED AND OF COMPOSITE CLASSES OF UNLIBERATED CHALCOPYRITE IN THE 17 MILL PRODUCTS.

PRODUCT.	Cu RECOVERY	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	90-100	70-100
1. Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	7.47	2.65	15.8	28.5	48.1	22.1	16.4	5.49	4.21
3. Cu-Roug Co.	103.1	107.5	98.3	7.28	229.6	78.9	103.6	109.2	106.7
4. Cu-Cl.1 Fd.	107.8	112.9	102.4	78.8	251.6	85.8	107.2	114.2	111.4
5. Cu-Cl.1 Co.	96.3	104.6	83.9	21.1	131.9	33.9	94.8	104.4	101.9
6. Cu-Cl.2 Co.	91.6	99.3	79.7	15.0	109.9	27.0	91.2	99.4	97.3
7. Zn-Roug Fd.	9.12	0.88	22.2	92.3	0.35	79.2	9.63	1.09	3.10
8. Zn-Roug Co.	4.23	0.18	10.4	54.9	44.9	41.7	2.42	0.48	0.95
9. Zn-Cl.1 Co.	5.84	0.37	14.6	67.4	46.3	47.4	5.82	1.61	2.36
10. Zn-Cl.2 Co.	3.47	0.04	8.53	47.6	34.6	35.5	1.61	0.03	0.59
11. Cu-Roug Tl.	15.8	3.38	36.1	113.5	38.2	95.0	25.2	6.12	6.94
12. Cu-Scav Tl.	8.36	0.73	20.3	84.9	-9.95	73.0	8.82	0.63	2.74
13. Cu-Cl.1 Tl.	11.5	8.27	18.6	57.8	119.7	51.9	12.5	9.80	9.41
14. Cu-Cl.2 Tl.	4.70	5.33	4.13	6.05	22.0	6.84	3.60	5.02	4.69
15. Zn-Cl.1 Tl.	0.76	0.15	1.89	7.27	10.3	6.02	0.81	0.46	0.36
16. Zn-Cl.2 Tl.	2.37	0.33	6.05	19.8	11.7	11.9	4.21	1.58	1.77
17. Zn-Roug Tl.	4.89	0.70	11.8	37.4	-44.6	37.5	7.21	0.60	2.15

(i.e. 30-70). In the latter case the negative values for the adjusted modal fraction of chalcopyrite were taken as zero, resulting in the '-0.00 %' of the Cu recovery being attributed to middling chalcopyrite in the Cu-scavenger and Zn-rougher tailings. Though this simplification is most probably erroneous, the amount of chalcopyrite in the middling classes is less than 5 %, and of which possibly only 50 % is lost from the Cu-circuit. By corollary, the 1.39 % of the Cu recovery in the Cu-concentrate attributed to middling chalcopyrite is correspondingly elevated. The behaviour of middling chalcopyrite is more readily determined by comparison of the sub-middling (i.e. 0-30) and the middling and sub-middling (i.e. 0-70) composite classes.

Thus, with the exception of the middling composite class, all the columns contain data values that were extracted from satisfactorily material balanced variables. However, each variable was balanced independently of the other, such that the variables do not necessarily cross-balance i.e. sum up to a logical total along each horizontal row, corresponding to a given product. In the feed for example, (1) the adjusted % unliberated (35.8 %) and the % free (60.6 %) sum to 96.4 %; (2) the recovery of Cu attributed to sub-middling (3.27 %), middling (1.27 %), sub-free (28.4 %) and free (60.6 %) chalcopyrite sums to 93.5 %; and (3) the recovery of Cu attributed to sub-middling plus middling (6.95 %) and sub-free plus free (90.4 %) chalcopyrite sums to 97.4 %; all of which should ideally sum to 100 %. Similarly, in the Cu-concentrate the recovery of Cu attributed to sub-middling (0.30 %), middling (1.39 %), sub-free (25.9 %) and free (60.2 %) chalcopyrite sums to 87.8 %; while the recovery of Cu attributed to sub-middling plus middling (1.88 %) and sub-free plus free (87.9 %) chalcopyrite sums to 89.8 %, both of which fall slightly short of the ideal of 91.6 % Cu recovery in this product. The totals in these products are on average depressed by approximately 4 % of the ideal total, suggesting relatively close convergence of the material balances on the true values, and hence acceptable crosss-balancing. However, one obvious deviation from acceptable cross-balancing is the greater percent of the Cu recovery attributed to the sub-free composite class rather than the sub-free plus free composite class in the Cu-scavenger concentrate (i.e. 4.68 and 3.82), the Cu-rougher tailings (i.e. 7.17 and 6.27), the Cu-scavenger tailings (i.e. 2.51 and 2.47) and the Zn-rougher tailings (i.e. 2.05 and 1.94). However, all the above products are known to contain negligible free chalcopyrite (See FREE GRAINS). Thus, it is assumed that the results for individual material balanced composite classes are more accurate (i.e. approach the true values), where a greater number of individual classes are included (i.e. 0-70 as opposed to the 30-70 composite classes), and hence where the counting statistics are improved.

From the first row of data in Table 29, corresponding to the float feed, 6.95 % of the Cu is attributed to middling and sub-middling chalcopyrite. An additional 90.38 % of the Cu is attributed to free and sub-free chalcopyrite, the remainder of 2.65 % of the Cu being unaccounted for on recombining the material balances. With respect to the middling and sub-middling classes, the value of 6.95 % is the most precise estimate of the Cu in the feed occurring in these classes, whereas the value of 1.27 % occurring solely in the middling classes is the most imprecise (i.e. poor material balance). Subtracting the value of 3.27 % for the sub-middling classes from that of the combined middling and sub-middling classes, a value of 3.68 % of the Cu occurring as middling chalcopyrite is generated. Thus, approximately 3.5 % of the Cu in the

feed occurs as middling chalcopyrite, of which approximately 50 % is recovered in the Cu-concentrate. Similarly (i.e. 2.60 - 1.22), approximately 1.4 % of the Cu in the feed reports to the tailings as middling chalcopyrite, while approximately 0.9 % (i.e. 2.47 - 1.56) of the Cu is recovered in the Zn-concentrate as middling chalcopyrite. Thus, regrinding of the Cu-1st cleaner tailings, which recovers approximately 1.7 % (i.e. 3.61 - 1.89) of the Cu as middling chalcopyrite, is unlikely to yield a pronounced improvement in the Cu recovery. Of the 3.27 % Cu due to middling chalcopyrite in the feed, 0.49 % is recovered in the Cu-concentrate, 1.56 % in the Zn-concentrate (principally as chalcopyrite-disease and diablastic sphalerite-chalcopyrite particles), and 1.22 % in the tailings. The sub-middling classes are not amenable to regrinding, as the individual fine-grained chalcopyrite grain forms less than 30 volume % of the particle, and would require excessive grinding of the particle for liberation. Thus, the 1.22 % Cu recovery in the tailings, attributed to sub-middling chalcopyrite, is essentially unrecoverable (See Fig. 9a).

Free and sub-free chalcopyrite comprise 90.38 % of the Cu in the feed. This constitutes excellent 'effective liberation', as these grains should behave as free chalcopyrite during flotation. Of this 97.3 % reports to the Cu-concentrate, such that 87.9 % of the Cu in the feed is recovered in the Cu-concentrate as free and sub-free (i.e. recoverable) classes of chalcopyrite. The remaining 2.47 % of the Cu in the feed occurring as free and sub-free chalcopyrite and which is not recovered in the Cu-concentrate, is recovered in the Zn-concentrate (0.53 %) and the tailings (1.94 %). Much of that reporting to Zn-concentrate consists of coarse-grained essentially free chalcopyrite, whose grain size is outside the optimum grain size range for flotation i.e. 5 to 100 um. Regrinding of the Cu-1st cleaner tailings, in which this fraction of chalcopyrite will likely tend to concentrate, may improve the recovery into the Cu-concentrate.

The approximately 2 % Cu recovery (i.e. 2.05 % for the 70-99.9 classes and 1.94 % for the 70-100 classes) in the tailings, attributed to free and sub-free chalcopyrite, corresponds to an assay of approximately 0.05 % Cu, or approximately 40 % of the Cu in the tailings. Thus, the loss of 2.15 % of the free and sub-free chalcopyrite to the tailings corresponds to a loss of approximately 2 % of the Cu in the feed. These constitute minimal losses of the recoverable classes of chalcopyrite, yet contribute to approximately 40 % of the Cu lost to the tailings. For the routine selective flotation of chalcopyrite, which normally achieves efficiencies in the upper 95 percent range for these classes of chalcopyrite (W. Petruk, pers. comm. 1987) these are generally acceptable losses. Thus, excellent efficiency is indicated by the flotation and recovery of 97.3 % of the recoverable classes of chalcopyrite.

The raw % unliberated chalcopyrite in various selected categories of particles are presented in Table 30. The data in the columns labelled 'Gang', 'Sulf', 'Py+Po' and 'Sphal' represent subsets of that in 'All', where the latter represents the modal percent of chalcopyrite occurring in particles containing any other mineral, and is therefore equivalent to the % unliberated chalcopyrite. Combining 'Gang' and 'Sulf' or 'Gang', 'Py+Po' and 'Sphal' yield values in excess of that in 'All', while combining 'Py+Po' and 'Sphal' yields values in excess of that in 'Sulf'. This apparent disparity results from the

TABLE 29

THE MODAL FRACTION OF THE Cu RECOVERY OCCURRING IN COMPOSITE CLASSES OF UNLIBERATED CHALCOPYRITE IN THE 17 MILL PRODUCTS.

PRODUCT.	Cu RECOVERY	ADJUSTED % FREE	ADJUSTED % LOCKED	LOCKED 0-99.9	COMPOSITE CLASSES OF UNLIBERATED GRAINS. 0-30	30-70	0-70	70-99.9	70-100	FREE 99.9-100
1. Feed	100.0	60.6	35.8	35.8	3.27	1.27	6.95	28.4	90.4	60.6
2. Cu-Scav Co.	7.47	21.5	75.9	5.67	0.94	0.61	1.54	4.68	3.82	1.61
3. Cu-Roug Co.	103.1	63.2	34.1	35.2	2.38	2.91	5.48	29.5	96.4	65.2
4. Cu-Cl.1 Fd.	107.8	63.5	34.0	36.7	2.58	3.19	5.96	30.5	100.6	68.5
5. Cu-Cl.1 Co.	96.3	65.8	31.1	30.0	0.69	1.67	2.35	26.9	92.1	63.4
6. Cu-Cl.2 Co.	91.6	65.7	31.1	28.5	0.49	1.39	1.88	25.9	87.9	60.2
7. Zn-Roug Fd.	9.12	5.83	86.8	7.93	3.01	0.01	5.50	2.74	2.80	0.53
8. Zn-Roug Co.	4.23	2.63	88.1	3.73	1.79	0.57	2.90	0.69	0.86	0.11
9. Zn-Cl.1 Co.	5.84	3.80	89.3	5.22	2.20	0.59	3.29	1.65	2.13	0.22
10. Zn-Cl.2 Co.	3.47	0.63	88.0	3.05	1.56	0.44	2.47	0.46	0.53	0.02
11. Cu-Roug Tl.	15.8	12.9	81.7	12.9	3.71	0.48	6.60	7.17	6.27	2.04
12. Cu-Scav Tl.	8.36	5.30	86.8	7.26	2.78	-0.00	5.07	2.51	2.47	0.44
13. Cu-Cl.1 Tl.	11.5	43.7	57.9	6.64	1.89	0.87	3.61	3.54	8.50	5.01
14. Cu-Cl.2 Tl.	4.70	68.7	31.4	1.48	0.20	0.28	0.48	1.02	4.24	3.23
15. Zn-Cl.1 Tl.	0.76	11.8	88.7	0.68	0.24	0.13	0.43	0.23	0.33	0.09
16. Zn-Cl.2 Tl.	2.37	8.45	91.4	2.17	0.65	0.15	0.83	1.20	1.60	0.20
17. Zn-Roug Tl.	4.89	8.62	85.9	4.20	1.22	-0.00	2.60	2.05	1.94	0.42

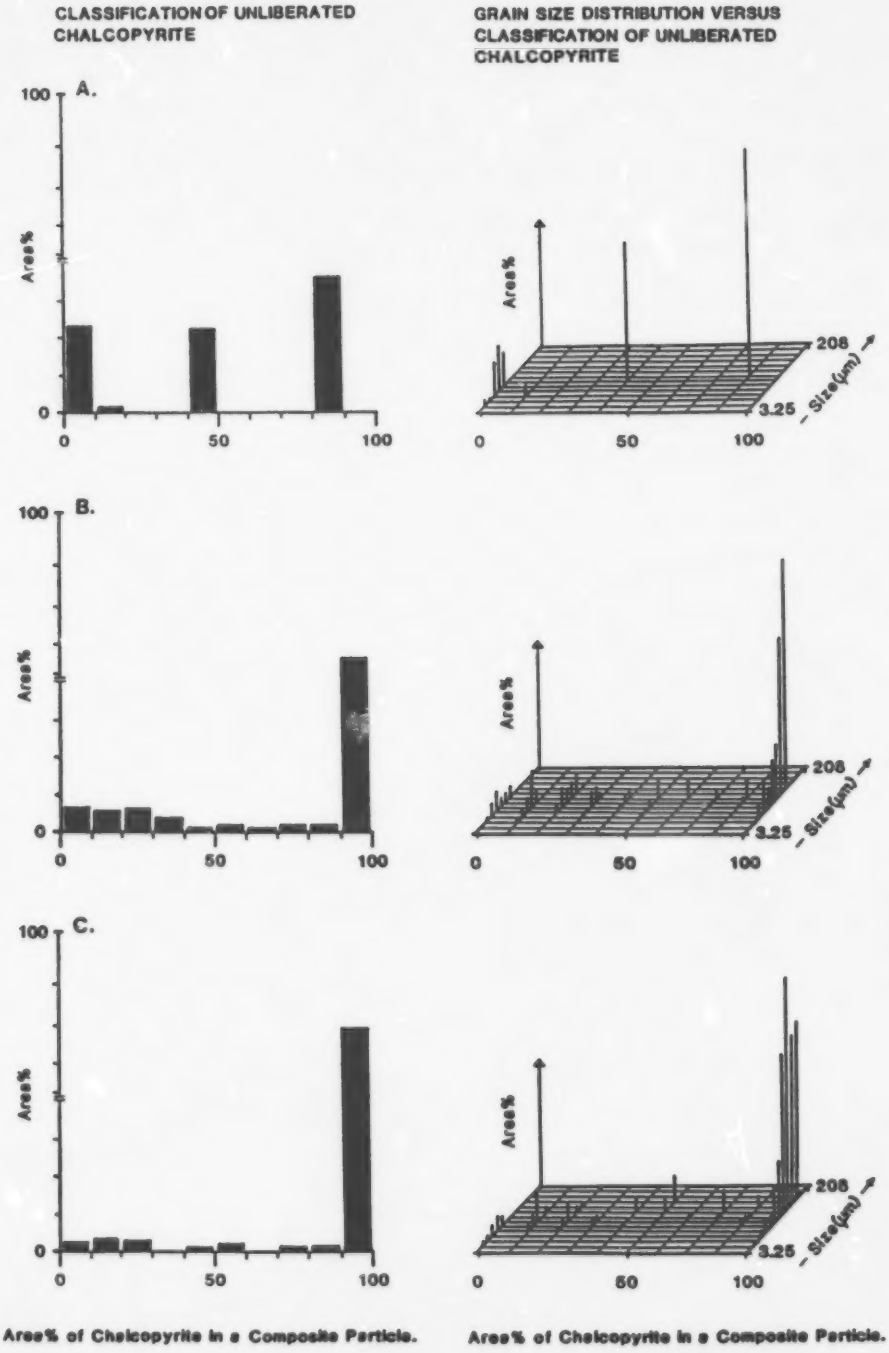


FIGURE 9.
PLOTS OF THE RAW CHALCOPYRITE CLASSIFICATION IN (A) THE TAILINGS, (B) THE Cu-1st CLEANER TAILINGS, AND (C) THE Cu-SCAVENGER CONCENTRATE.

unliberated chalcopyrite occurring in particles that may contain grains of one or more other minerals, and thus be identified as occurring in several of the selected categories. Thus, the greater the difference between the % unliberated and these totals, which might be expected to equal the former, the greater the complexity of these particles containing unliberated chalcopyrite (i.e. ternary and quatenary particles).

In the feed, 43.6 % of the chalcopyrite is unliberated with 85 % (i.e. 37.0 %) of this chalcopyrite locked with gangue minerals, 33 % (i.e. 14.3 %) locked with pyrite or pyrrhotite and 28 % (i.e. 12.3 %) locked with sphalerite. The predominance of gangue minerals (i.e. silicates and carbonates) as the accompanying minerals in particles containing unliberated chalcopyrite is reflected in all the unliberated chalcopyrite in the tailings being locked with gangue (i.e. 96.1 %), with a lesser amount (i.e. 69.0 %) locked with pyrite or pyrrhotite. This is largely substantiated by petrographic observation.

In the Cu-scavenger concentrate, 81.6 % of the chalcopyrite is unliberated with roughly equivalent amounts locked with gangue (42.8 %), pyrite or pyrrhotite (47.8 %), and sphalerite (45.0 %). The abundance of chalcopyrite-sphalerite particles in the Cu-scavenger concentrate supports the earlier made contention that the sphalerite in this product is largely unliberated, and being locked with chalcopyrite is recovered by the 'hard pulling' Cu-scavenger. Regrinding to liberate the chalcopyrite from the gangue, pyrite, pyrrhotite and particularly sphalerite, may be warranted on this small stream (i.e. product weight of 1.92%). The above also tends to explain the high concentration factors for Zn and sphalerite in the Cu-scavenger concentrate, which is not interpreted to be the principal source of the free sphalerite that reports to the Cu-concentrate. The free and sub-free sphalerite that is recovered in the Cu-concentrate is interpreted to be initially floated principally at the Cu-rougher stage. This is reflected in the proportionally lower contribution of chalcopyrite-sphalerite particles to the % unliberated chalcopyrite in the Cu-rougher concentrate (i.e. 8.9 % out of 39.6 %).

In the Zn-2nd cleaner concentrate, 98.2 % of the chalcopyrite occurs in particles containing sphalerite, and principally reflects chalcopyrite-disease and diablastic chalcopyrite-sphalerite particles, the former being principally comprised of sub-middling, and hence unrecoverable grains of chalcopyrite (See Fig. 8b).

The Cu-1st cleaner tailings contains 56.2 % unliberated chalcopyrite with roughly equivalent amounts (i.e. approximately 20 %) locked with gangue and with sphalerite, and an even higher amount (i.e. 29 %) locked with pyrite and/or pyrrhotite. Regrinding of this product is recommended, as it contains 51.9 % of the middling and sub-middling chalcopyrite concentrated in 5.67 % of feed stream flow rate, and thus provides the highest opportunity for improved Cu recovery through enhanced chalcopyrite liberation. In that one third of the unliberated chalcopyrite is also locked with sphalerite, suggests that such a regrind will also improve the liberation and selective flotation of sphalerite, and hence Zn recovery to the Zn-concentrate.

Both the Cu-1st cleaner tailings and the Cu-scavenger concentrate offer the highest potential for improved liberation of chalcopyrite (and sphalerite

TABLE 30

CONDENSED RAW LIBERATION DATA FOR SELECTED CATEGORIES OF UNLIBERATED CHALCOPYRITE IN THE 17 MILL PRODUCTS.

(in Area % chalcopyrite in each category).

PRODUCT.	FREE GRAINS (% Free)	UNLIBERATED GRAINS				
		All.	Gang.	Sulf.	Py+Po.	Sphal.
1. Feed.	56.4	43.6	37.0	17.3	14.3	12.3
2. Cu-Scav Co.	18.4	81.6	42.8	68.5	47.8	45.0
3. Cu-Roug Co.	60.4	39.6	20.1	26.5	19.3	8.9
4. Cu-Cl.1 Fd.	64.1	35.9	15.1	24.5	14.7	9.0
5. Cu-Cl.1 Co.	65.7	34.3	15.8	26.4	12.1	16.1
6. Cu.Cl.2 Co.	75.4	24.6	17.8	15.4	10.0	5.7
7. Zn-Roug Fd.	3.4	96.6	----	----	----	----
8. Zn-Roug Co.	15.6	84.4	30.3	73.6	45.0	44.1
9. Zn-Cl.1 Co.	13.5	86.5	36.6	71.3	45.8	43.0
10. Zn-Cl.2 Co.	0.3	99.7	41.4	99.7	68.0	98.2
11. Cu-Roug Tl.	27.0	73.0	63.7	36.5	24.8	30.4
12. Cu-Scav Tl.	21.9	78.1	54.2	74.3	59.6	63.8
13. Cu-Cl.1 Tl.	43.8	56.2	18.2	45.1	29.1	20.6
14. Cu-Cl.2 Tl.	68.7	31.3	11.4	23.0	16.3	6.3
15. Zn-Cl.1 Tl.	11.5	88.5	78.0	76.0	43.9	51.3
16. Zn-Cl.2 Tl.	7.3	92.7	63.1	79.7	52.7	49.1
17. Zn-Roug Tl.	3.9	96.1	96.1	69.0	69.0	----

Note : '----' denotes data lost during processing.

locked with chalcopyrite) by regrinding. This is based on their product weights or stream flow rates (i.e. 5.67 and 1.93, respectively) and the abundance of unliberated chalcopyrite (i.e. 51.9 % and 22.1 % of the middling and sub-middling chalcopyrite in the feed, respectively, See Figs. 9b and 9c). In addition, both streams, but particularly the Cu-scavenger concentrate contain abundant sphalerite that has been (at least initially) floated because it is locked with chalcopyrite. As both streams are returned to the Cu-rougher stage, regrinding of a combined stream with a flow rate of 7.60, prior to the Cu-rougher stage, may warrant investigation. Such a stream would contain 74.1 % of the middling and sub-middling chalcopyrite, much of which is locked with sphalerite (i.e. approximately 30 %), and thus the two principal objectives of regrinding.

At present 60 % (split stream) of the Cu-rougher tailings is reground, and this constitutes a stream flow rate of approximately 57. This reduced stream contains 57 % (i.e. 60 % of 95 %) of the middling and sub-middling chalcopyrite, and 30 % of the unliberated chalcopyrite is locked with sphalerite. Regrinding of the combined Cu-1st cleaner tailings and Cu-scavenger concentrate streams offers a much more selective target for regrinding, in that essentially comparable material is concentrated in a much smaller stream, permitting a much lower grain size distribution and hence higher liberation to be achieved in the reground product.

SPHALERITE

The raw liberation and classification data for sphalerite in each of the 25 mill product samples (compiled from the raw % free and raw 'sphalerite Vs all' classifications) are given in Table 8 of appendix B, and are condensed for the 17 mill products in Table 31. Each of the values in Table 31 were multiplied by the corresponding adjusted Zn assay for that product to yield the raw modal fraction of the adjusted Zn assay occurring as (1) unliberated grains, (2) as unliberated grains in each class, and (3) as free grains. This mineral-metal data is given in Table 9 of appendix B. The data for each of the ten classes of unliberated sphalerite in this table formed the input data for processing by MATBAL. The purpose of this was to generate adjusted mineral-metal data values from which the adjusted modal amounts of unliberated sphalerite in each class could be extracted (by dividing by the corresponding adjusted Zn assay), and the recoveries of each class of unliberated sphalerite in each product was simultaneously determined.

From the raw data on the first row of Table 31, corresponding to the float feed, it is apparent that while only 41.8 % of the sphalerite occurs as free grains, another 48.2 % occurs in the 70-99.9 or sub-free classes (See Fig. 10a). Thus, in the raw data 90.0 % of the sphalerite occurs in the free and sub-free or recoverable classes of sphalerite. The remaining 10.0 % of the sphalerite is split between the middling classes (8.86 %) and the sub-middling classes (1.10 %). The counting statistics for sphalerite in the individual classes therefore become progressively poorer through the middling and sub-middling classes. Consequently, the material balance calculations are expected to be correspondingly less precise and accurate, yielding unsatisfactory results. Similarly, products such as the tailings that are highly depleted in sphalerite exhibit erratic classifications due to poor counting statistics.

Th calculated recovery of the free, of the unliberated and of each class of the sphalerite classification in each product is presented in Table 32. The data in the columns labelled 'Zn Recovery', '% Locked' and '% Free' and which have been introduced in previous sections, are presented here for ease of comparison. Six of the individual classes generated good material balances, whereas the 80-90 class generated a negative value of -5.42 % in the tailings, and hence an unsatisfactory balance. The 40-50, 20-30 and 10-20 classes each generated at least one negative recovery value in excess of -75 %, and are therefore rejected. This is principally a product of poor counting statistics within each class of the unliberated sphalerite. The erratic nature of the material balances for each of the individual classes of sphalerite prohibits useful discussion of these results.

The adjusted modal fractions of the Zn assay occurring in each class of unliberated sphalerite were combined into several composite classes, each with relatively predictable and uniform flotation characteristics (See Footnote 2). These values are presented in Table 10 of appendix B, and comprise the input data to MATBAL, from which satisfactory material balances were generated. The influence of poor counting statistics in the individual classes are reduced by combining the individual into composite classes. The calculated recovery of the modal fractions of the Zn assay occurring in the composite classes are given in Table 33.

As stated earlier, 90.7 % of the free and 72.0 % of the unliberated spghlerite are recovered in the Zn-concentrate (See Fig. 10b), resulting in a corresponding Zn reovery of 82.5 %. The major losses of sphalerite are to the Cu-concentrate where 9.29 % of the free and 13.2 % of the unliberated are recovered, resulting in a corresponding Zn loss of 11.0 %. The loss of 14.8 % of the unliberated sphalerite to tailings, accounts almost completely for the 6.54 % Zn loss, in that only 0.01 % of the free sphalerite is lost to the tailings. While, the recovery of free sphalerite is greater than that of unliberated sphalerite in the Zn-concentrate, the reverse is true in the Cu-concentrate. This suggests that a component of the sphalerite reporting to the Cu-concentrate is locked with chalcopyrite, and is therefore floated in the Cu-circuit, due to a liberation-dependent mechanism.

While the recovery/loss of free sphalerite into the Cu-concentrate is 9.29 %, that of essentially free sphalerite (i.e. 90-100 classes) is 10.1 %, and that of sub-free sphalerite is 12.1 %. The recovery of these classes combined (i.e. 70-100 or recoverable classes of sphalerite) is 10.3 %. Furthermore, the recovery in the Cu-concentrate of middling sphalerite is 19.5 %, and 22.5 % of submiddling sphalerite, demonstrating a progressive increase in the recovery sphalerite into the Cu-concentrate with the degree of locking (Compare Figs. 10a and 10c). However, because middling and sub-middling sphalerite comprise only 8.40 % of the sphalerite in the feed, the increased recovery of these classes, does not substantially alter the overall recovery of unliberated sphalerite i.e. 10.3 % for the 70-100 classes, and 13.2 % for all unliberated sphalerite (0-99.9 classes). In as much as the recovery of sphalerite increases with its degree of locking (presumably with chalcopyrite), the fact that 9.29 % of the free sphalerite is recovered indicates that the bulk of the sphalerite in the Cu-concentrate is liberation independent. Assuming the mechanism by which this component of the sphalerite is recovered is liberation-independent i.e. possibly activation, and that

TABLE 31

CONDENSED RAW LIBERATION AND CLASSIFICATION DATA FOR SPHALERITE IN THE 17 MILL PRODUCTS.

PRODUCT.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.									% FREE
		0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	58.2	0.40	0.60	0.10	1.23	1.91	2.30	3.42	4.65	2.22	41.3	41.8
2. Cu-Scav Co.	85.6	1.79	1.10	1.31	3.10	2.62	1.79	1.57	7.06	14.2	50.8	14.4
3. Cu-Roug Co.	60.3	1.57	1.60	1.16	2.79	1.93	3.33	5.53	4.82	8.56	29.0	39.7
4. Cu-Cl.1 Fd.	66.0	0.74	0.82	1.39	1.47	1.30	6.02	7.05	6.96	14.1	26.4	34.0
5. Cu-Cl.1 Co.	55.1	3.27	1.49	0.61	2.06	1.61	0.37	2.49	4.52	7.35	31.3	44.9
6. Cu-Cl.2 Co.	50.3	1.34	1.35	1.29	1.25	2.60	1.99	1.86	2.73	9.02	26.8	49.7
7. Zn-Roug Fd.	42.6	1.41	0.17	0.78	0.34	1.04	2.32	0.27	3.63	1.22	31.3	57.4
8. Zn-Roug Co.	41.3	0.41	0.67	0.99	0.58	1.09	1.08	1.35	1.95	5.42	27.8	58.7
9. Zn-Cl.1 Co.	38.2	0.73	1.04	0.76	0.39	1.34	1.53	0.51	1.26	2.70	27.9	61.8
10. Zn-Cl.2 Co.	36.8	0.17	0.48	0.55	0.83	0.72	2.20	0.70	1.55	5.42	24.3	63.2
11. Cu-Roug Tl.	57.0	0.39	2.60	0.85	0.40	0.00	0.22	2.61	5.11	14.6	30.2	43.0
12. Cu-Scav Tl.	49.6	0.86	1.63	0.93	0.67	2.02	1.15	2.26	2.81	16.7	20.6	50.4
13. Cu-Cl.1 Tl.	63.4	0.88	1.50	0.99	0.87	1.28	1.51	3.36	9.13	10.4	33.6	36.6
14. Cu-Cl.2 Tl.	34.4	0.45	0.51	0.29	0.88	1.70	1.38	2.81	3.13	6.66	16.6	65.6
15. Zn-Cl.1 Tl.	78.3	27.7	19.5	6.59	6.34	2.29	2.01	1.80	0.15	8.92	3.16	21.7
16. Zn-Cl.2 Tl.	87.7	13.5	20.1	11.2	9.51	9.32	13.5	5.16	2.18	1.33	1.97	12.3
17. Zn-Roug Tl.	100.0	31.3	19.0	0.00	0.00	0.00	0.00	5.66	0.00	44.2	0.00	0.00

TABLE 32

RECOVERY OF THE FREE, UNLIBERATED AND EACH CLASS OF UNLIBERATED SPHALERITE IN THE 17 MILL PRODUCTS.

PRODUCT.	Zn RECOVERY	% LOCKED (0-99.9)	CLASSIFICATION OF UNLIBERATED GRAINS. 0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	% FREE (99.9-100)
1. Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	8.12	15.8	26.7			40.4		18.1	17.4	33.3	51.3	16.3	2.36
3. Cu-Roug Co.	26.8	35.9	53.5			69.0		61.1	112.9	84.6	106.2	30.8	19.9
4. Cu-Cl.1 Fd.	30.0	38.3	56.2	N	N	75.3	N	67.0	124.5	90.1	114.8	32.8	23.8
5. Cu-Cl.1 Co.	14.2	15.6	31.3	O	O	44.3	O	13.9	40.5	23.9	45.7	14.2	13.2
6. Cu-Cl.2 Co.	11.0	13.2	28.7			38.0		7.96	28.9	18.4	37.1	12.1	9.29
7. Zn-Roug Fd.	90.7	89.7	141.5	B	B	83.9	B	98.3	74.6	81.6	68.6	88.0	91.4
8. Zn-Roug Co.	84.2	74.9	92.4	A	A	83.2	A	97.7	44.2	81.4	74.0	88.0	91.4
9. Zn-Cl.1 Co.	87.6	81.5	144.8	L	L	101.1	L	214.9	69.1	86.9	71.3	88.2	91.9
10. Zn-Cl.2 Co.	82.5	72.0	22.2	A	A	61.3	A	91.5	40.8	81.4	68.3	87.8	90.7
11. Cu-Roug Tl.	97.3	102.6	98.1	N C	N C	102.4	N C	110.2	88.6	114.9	114.2	104.1	93.1
12. Cu-Scav Tl.	89.0	86.8	71.3	E	E	62.0	E	92.0	71.1	81.6	62.9	87.8	90.7
13. Cu-Cl.1 Tl.	15.9	22.7	24.9	.	.	31.0	.	53.1	84.0	66.2	69.1	18.6	10.6
14. Cu-Cl.2 Tl.	1.18	2.41	2.68			6.31		5.92	11.6	5.50	8.57	2.03	3.91
15. Zn-Cl.1 Tl.	1.69	2.88	70.1			22.0		6.26	3.42	0.00	5.74	0.22	0.73
16. Zn-Cl.2 Tl.	5.08	9.50	122.5			39.9		123.4	28.4	5.48	2.98	0.38	1.24
17. Zn-Roug Tl.	6.54	14.8	49.1			0.71		0.59	30.4	0.18	-5.42	0.13	0.01

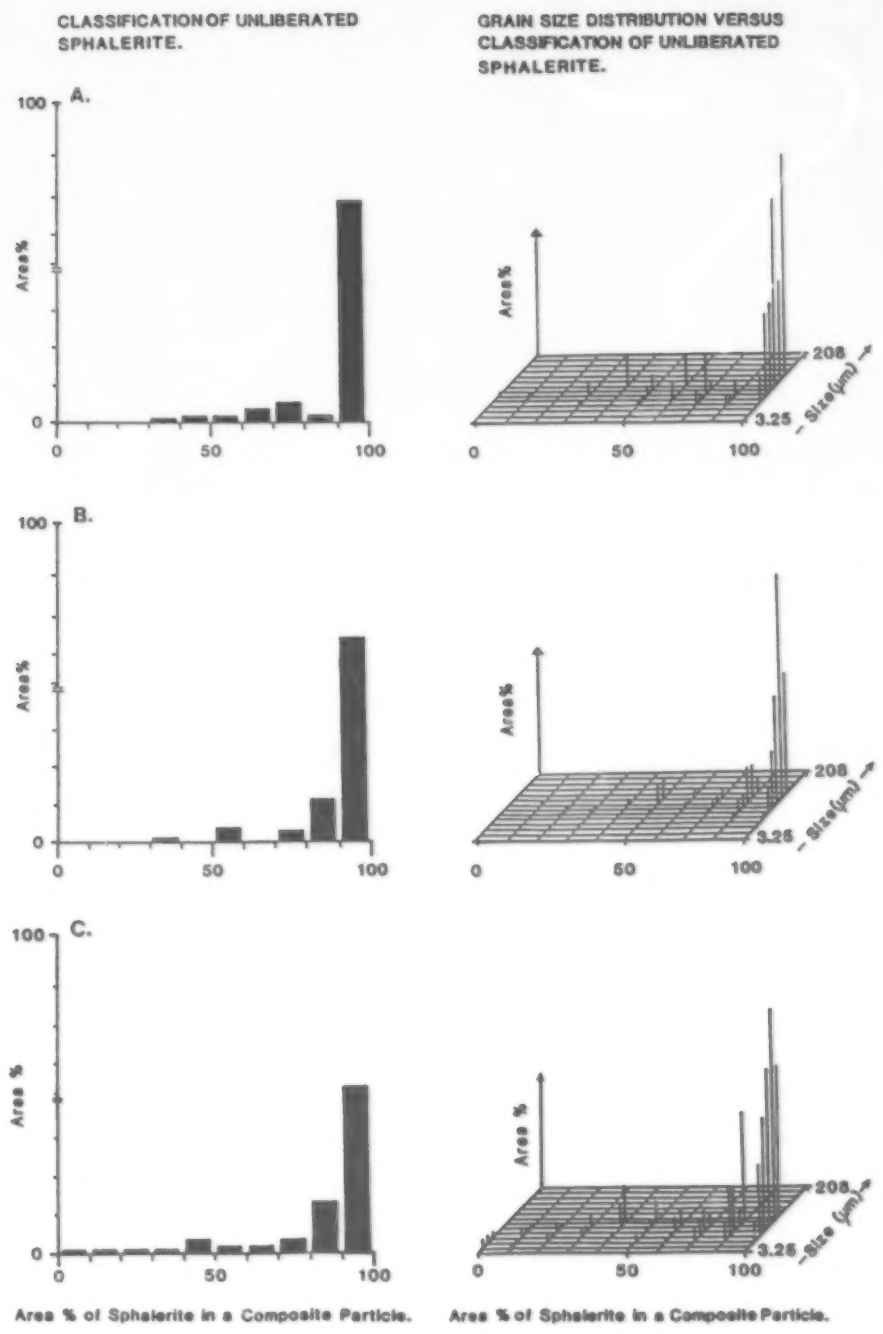


FIGURE 10.
PLOTS OF THE RAW SPHALERITE CLASSIFICATION IN (A) THE FEED, (B) THE Zn-2nd CLEANER CONCENTRATE, AND (C) THE Cu-2nd CLEANER CONCENTRATE.

therefore unliberated sphalerite is similarly affected regardless of the degree of locking, then approximately 4 % (i.e. 13.2 % - 9.29 %, or one third) out of the 13.2 % sphalerite is recovered due to locking, most probably with chalcopyrite.

The losses to the Cu-concentrate of 9.29 % of the free, 12.1 % of the sub-free, 19.5 % of the middling and 22.5 % of the sub-middling sphalerite, dictate that the Cu-scavenger tailings, the feed to the Zn-circuit contains 90.7 % of the free, 87.9 % of the sub-free, 80.5 % of the middling and 77.5 % of the sub-mddling sphalerite. Thus, despite recoveries in the Zn-concentrate of only 90.7 % of the free, 80.3 % of the sub-free, 65.0 % of the middling and 38.7 % of the sub-middling sphalerite, these correspond to recoveries of 100 %, 91.4 %, 80.7 % and 49.9 %, respectively of the corresponding composite classes of sphalerite that enter the Zn-circuit. The recovery of the free and sub-free sphalerite that enters the Zn-circuit, and is thus available for flotation, is 96.7 % (i.e. 86.7 % of 89.7), which is even greater than the recovery of the corresponding composite classes of chalcopyrite in the Cu-concentrate.

The recovery of 65 % of the middling sphalerite in the Zn-concentrate, and which corresponds to a recovery of 80.7 % of the middling sphalerite entering the Zn-circuit, is most interesting. While a particle consisting of 50 % sphalerite and 50 % chalcopyrite may not be recovered in the Cu-concentrate, it will clearly be recovered in the Zn-concentrate, as initially the sphalerite should behave inertly, whereas after activation with $CuSO_4$ the particle should be completely hydrophobic and flotable. Thus, middling sphalerite-chalcopyrite particles have a higher probability of being recovered in the Zn-concentrate. Consequently, if a large fraction of middling sphalerite is locked with chalcopyrite, as is indicated petrographically and by image analysis, the recovery of middling sphalerite will be enhanced, and proportionally exceed that of middling chalcopyrite (regardless of the accompanying mineral in the particle) in the Cu-concentrate (i.e. approximately 50 % of the middling chalcopyrite is recovered in the Cu-concentrate).

As stated above, the principal losses of sphalerite occur in the Cu-concentrate, while only 6.54 % of the Zn, and hence sphalerite is lost to the tailings. The loss of 0.01 % of the free sphalerite in the feed to the tailings further indicates the efficient operation of the Zn-circuit. However, 7.55 % of the sub-free sphalerite is lost to the tailings, and is comparable to the 7.21 % loss of sub-free chalcopyrite to the tailings. This results in a 3.01 % loss of recoverable classes of sphalerite, and is thus somewhat greater than the 2.15 % loss of recoverable classes of chalcopyrite to the tailings. Nonetheless, a loss of 3.01 % of the recoverable classes of sphalerite from a feed (i.e. Cu-scavenger tailings) containing 89.7 % of the same, implies the Zn-circuit is operating at 96.6 % efficiency for the recovery of these recoverable classes of sphalerite. The Zn-circuit is therefore closely approaching the 97.3 % efficiency of the Cu-circuit for the recovery of recoverable classes of chalcopyrite. The sub-free sphalerite lost to the tailings consists principally of sphalerite-pyrite particles, as observed petrographically.

The combined middling and sub-middling classes (i.e. 0-70) exhibit 40.6 % losses to the tailings. This conflicts with the separate material balances of the combined middling classes (15.4 %) and the combined sub-middling classes (38.8 %). This is particularly true as the raw data for the feed indicates that the modal amounts of middling sphalerite are approximately 8 times greater than that of sub-middling sphalerite, and therefore should yield a weighted average for the combined classes (i.e. 0-70) of close to the 15.4 % loss. As the raw data for the combined middling and sub-middling classes contains a greater number of measurements, it is assumed that the material balances generated recovery/loss of 40.6 % is more accurate than those for the separate middling and sub-middling composite classes. Accepting this, then the recovery of middling sphalerite to the tailings should approach 40 %, and the recovery in the Zn-concentrate, which is also anomalously high (with respect to the combined 0-70 classes), should be correspondingly depressed to approximately 45 %. Thus, it is interpreted that approximately 40 % of both the middling and sub-middling classes are lost to the tailings. However, a loss of approximately 40 % of 8.86 % (raw data) of the sphalerite in the feed occurring as middling sphalerite, constitutes losses of approximately 3.5 % of the total of the 6.54 % Zn losses to the tailings, and hence approximately 55 % of the sphalerite in the tailings.It is therefore of generally comparable magnitude, in terms of modal amounts of sphalerite as that lost in the form of recoverable classes of sphalerite.

Losses of approximately 40 % of the middling sphalerite to the tailings, constituting approximately 3.5 % of the sphalerite in the feed, as well as the 19.5 % loss to the Cu-concentrate, indicate an area for improved Zn-recovery through regrinding of that fraction of sphalerite. The tentatively proposed regrinding of the combined Cu-1st cleaner tailings and the Cu-scavenger concentrate, which contain 29.7 % and 15.5 %, respectively of the middling sphalerite, is expected to yield small but improved Zn recovery (See Figs. 11b and 11c). The Cu-rougher tailings contains 96 % of the middling sphalerite, of which 58 % is reground (i.e. 60 % of 96). To regrind a stream containing 45 %, as opposed to 58 % of the middling sphalerite, with a flow rate and hence dilution of approximately one order of magnitude smaller, would permit a much lower grind and hence liberation to be achieved for the same grinding costs. Such a regrind should improve on the losses of sphalerite to the Cu-concentrate as well as the tailings. However, the behaviour of recoverable classes of sphalerite which comprise the bulk of the sphalerite in the Cu-concentrate might not be expected to be affected by regrinding; the mechanism of flotation is largely liberation-independent. The recoverable classes of sphalerite lost to the tailings, and which principally consists of sphalerite-pyrite particles, and which are not expected to report to the Cu-scavenger concentrate or Cu-1st cleaner tailings are unlikely to be addressed by such a regrind.

The recovery of the adjusted Zn assay in terms of the adjusted modal fraction of sphalerite as free grains, unliberated grains, and in composite classes of unliberated grains in each product are presented in Table 34. These values were generated by multiplying the Zn recovery in a product by the adjusted modal fraction of sphalerite occurring in the specific composite class of unliberated and/or free sphalerite. Thus, the value of 9.40 % under '70-100' grains in the Cu-2nd cleaner concentrate implies that 9.40 % out of 11.0 % Zn recovery (i.e. 86 %) is attributable to free and sub-free or

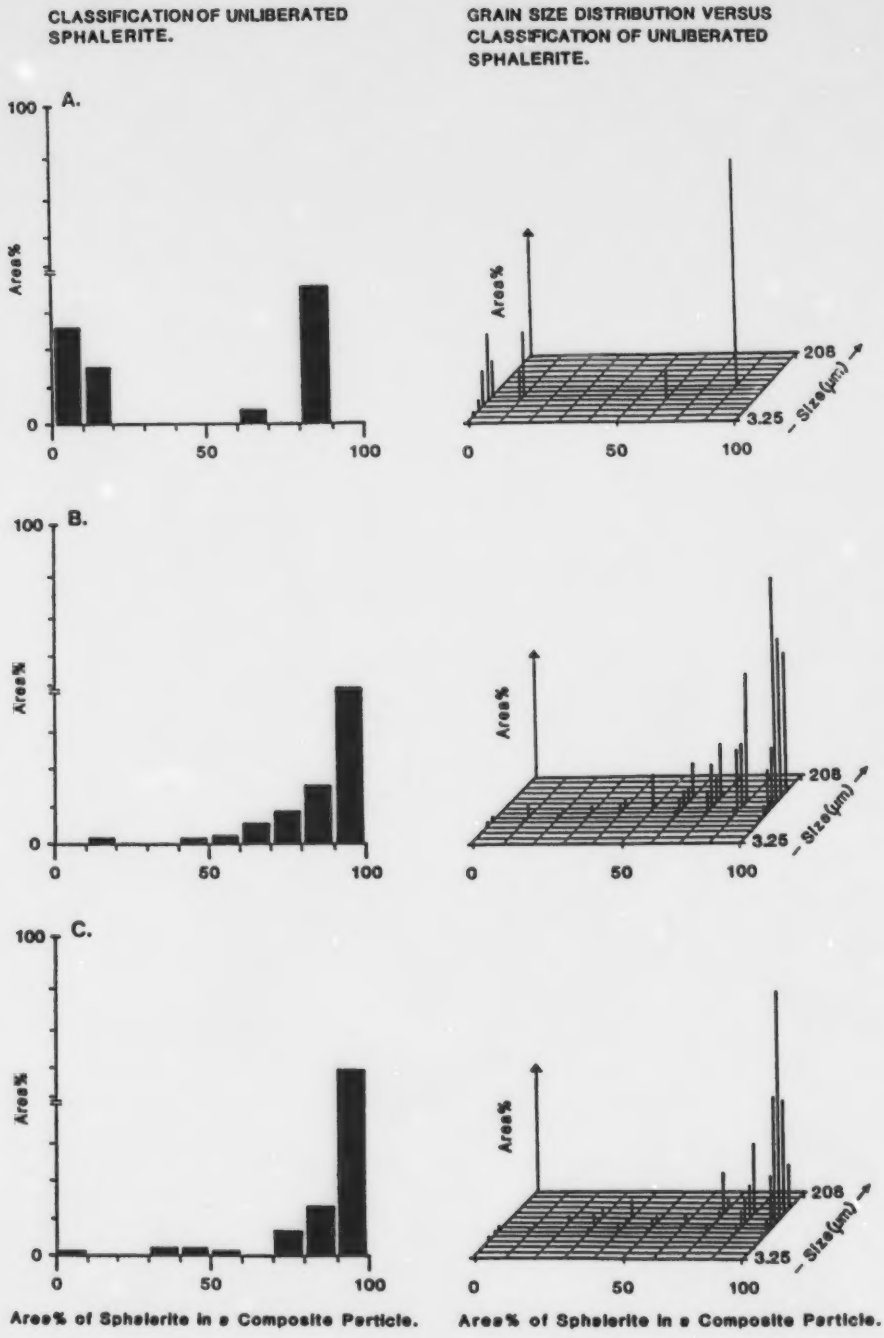


FIGURE 11.
PLOTS OF THE RAW SPHALERITE CLASSIFICATION IN (A) THE TAILINGS, (B) THE Cu-1st CLEANER TAILINGS, AND (C) THE Cu-SCAVENGER CONCENTRATE.

TABLE 33

RECOVERY OF THE FREE, UNLIBERATED AND OF COMPOSITE CLASSES OF UNLIBERATED SPHALERITE IN THE 17 MILL PRODUCTS.

PRODUCT.	Zn RECOVERY	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	90-100	70-100
1. Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	8.32	2.36	15.8	19.8	15.5	13.0	16.4	6.78	7.89
3. Cu-Roug Co.	26.8	19.9	35.9	51.3	49.3	40.6	33.5	22.9	24.9
4. Cu-Cl.1 Fd.	30.0	23.8	38.3	53.5	53.9	43.7	35.8	26.1	28.2
5. Cu-Cl.1 Co.	14.2	13.2	15.6	24.6	24.2	20.4	14.4	13.3	13.5
6. Cu-Cl.2 Co.	11.0	9.29	13.2	22.5	19.5	17.3	12.1	10.1	10.3
7. Zn-Roug Fd.	90.7	91.4	89.7	127.0	85.2	95.8	88.4	90.5	90.4
8. Zn-Roug Co.	84.2	91.4	74.9	88.2	69.7	55.2	80.9	90.5	87.4
9. Zn-Cl.1 Co.	87.6	91.9	81.5	139.1	98.3	82.4	81.1	90.8	87.7
10. Zn-Cl.2 Co.	82.5	90.7	72.0	38.7	65.0	42.1	80.3	89.9	86.7
11. Cu-Roug Tl.	97.3	93.1	102.6	97.3	96.0	95.7	104.3	96.7	97.6
12. Cu-Scav Tl.	89.0	90.7	86.8	77.5	80.5	82.7	87.9	89.9	89.7
13. Cu-Cl.1 Tl.	15.9	10.6	22.7	28.8	29.7	23.3	21.4	12.8	14.7
14. Cu-Cl.2 Tl.	3.18	3.91	2.41	2.15	4.62	3.06	2.27	3.23	3.18
15. Zn-Cl.1 Tl.	1.70	0.73	2.88	49.5	4.69	13.1	0.55	0.53	0.63
16. Zn-Cl.2 Tl.	5.08	1.24	9.50	100.3	33.3	40.3	0.75	0.91	0.99
17. Zn-Roug Tl.	6.54	0.01	14.8	38.8	15.4	40.6	7.55	0.00	3.01

recoverable classes of sphalerite. As all the composite classes generated good material balances (See Table 33) with the possible exception of the middling classes (See above), all the columns in Table 34 contain data values that were extracted from satisfactorily material balanced variables.

Nonetheless, as each variable was balanced independently of the others, and thus independently approached the true values, the variables do not necessarily cross-balance i.e. sum up to a logical total along each horizontal row corresponding to a given product. In the feed for example: (1) the adjusted % unliberated (46.1 %) and the % free (50.5 %) sum to 96.6 %; (2) the recovery of Zn attributed to sub-middling (1.86 %), middling (4.56 %), sub-free (37.6 %) and free (50.5 %) sphalerite sum to 94.5 %; and (3) the recovery of Zn attributed to sub-middling plus middling (8.40 %) and sub-free plus free (91.5 %) sum to 99.9 %, all of which should ideally sum to 100.0 %. These totals for the feed are on average depressed by approximately 3 %, suggesting relatively close convergence of the material balances on the true values, and hence acceptable cross-balancing. It is noteworthy, that the values for composite classes that incorporate a greater number of the individual classes, and hence number of measurements, yield cross-balances that closely approach the ideal totals i.e. 99.9 % for the sum of the middling plus sub-middling, and the free plus sub-free composite classes. Several deviations from acceptable cross-balancing are observed. The totals for the sub-middling and middling composite classes exceeded that for the combined sub-middling plus middling composite class in the Zn-rougher concentrate (i.e. $1.64 + 3.18 > 4.63$), the Zn-1st cleaner concentrate (i.e. $3.45 + 4.49 > 6.92$), and the Zn-2nd cleaner concentrate (i.e. $0.72 + 2.97 > 3.53$).

The values for the tailings also exhibit poor cross-balancing, reflecting the paucity of sphalerite in this sample, the poor counting statistics within each composite class, and the consequently high uncertainty in the raw modal amount of sphalerite in each class for the tailings. Though the value of 2.83 for the sub-free exceeds that of 2.75 for the sub-free plus free, the lack of free sphalerite in this sample implies that this small deviation is still within the acceptable ± 5 % uncertainty limits. In contrast, the sum of the sub-middling (0.72) and middling (0.71) is approximately 40 % of the value of 3.41 % for the combined sub-middling plus middling composite class. It is also noteworthy that the sum of the sub-middling (1.86) and middling (4.56) sphalerite in the feed falls short of that for the corresponding combined composite class (i.e. 8.40). It is interpreted that the paucity of sphalerite, and hence poor counting statistics in these separate composite classes, and particularly in samples with low modal abundances of these classes of sphalerite (e.g. the tailings) resulted in material balances that deviate significantly from the true values, with resultant poor cross-balancing. This is reflected in the adjustments made by MATBAL to the modal fraction of the Zn assay occurring as sub-middling sphalerite in the tailings (i.e. 0.100 adjusted to 0.021), and to the modal fraction of the Zn assay occurring as middling sphalerite in the feed (i.e. 0.230 adjusted to 0.120, See Table 10 appendix B).

From the first row of data in Table 34, corresponding to the float feed, 8.40 % of the Zn is attributed to middling and sub-middling sphalerite. Additionally 91.5 % of the Zn is attributed to free and sub-free sphalerite, and combined accounts for 99.9 % of the Zn in the feed. With respect to the

middling and sub-middling composite classes, the value of 8.40 % is the most precise estimate of the Zn in the feed occurring in these combined classes. The values of 1.86 % and 4.56 % for the separate sub-middling and middling composite classes, respectively are less precise, as is indicated by poor cross-balancing. It is interpreted that the true values more closely approach 2.0 % and 6.50 %, respectively. Assuming the latter to be true, then 2.0 % of the sphalerite in the feed is essentially unrecoverable, occurring in the sub-middling composite class, and will report to an end-product based on the flotation characteristics of the accompanying mineral(s) in the host particles. This 2.0 % fraction of the sphalerite is not amenable to regrinding. Approximately one third of this composite class is presently recovered in the Zn-concentrate, constituting a 0.72 % Zn recovery. The composite middling classes, which comprise 4.56 % (calculated) to 6.50 % (interpreted) of the sphalerite in the feed, are principally recovered in the Zn-concentrate, and constitutes a Zn recovery of approximately 2.97 % (calculated) to 3.5 % (interpreted). The Cu-concentrate and the tailings recover approximately 20 % and 30 %, respectively of the middling sphalerite, and constitutes a Zn recovery of approximately 1.6 % (calculated) to 2.5 % (interpreted). Thus, between 1.6 % and 2.5 % of the Zn in the feed occurring as middling sphalerite is lost, and is amenable to regrinding and consequent improved sphalerite liberation and Zn recovery. The combined Cu-1st cleaner tailings (1.36 %) and Cu-scavenger concentrate (0.71 %) recovers 2.07 % (calculated) to 2.5 % (interpreted) of the Zn in the feed occurring as middling sphalerite (See Figs. 11b and 11c). Regrinding of this stream is unlikely to yield significantly enhanced sphalerite liberation and hence a pronounced (i.e. 1 or 2 %) improvement in Zn recovery. However, assuming that activation or poor froth formation is the mechanism by which approximately 10 % of the sphalerite in the feed is recovered in the Cu-concentrate, and that this is liberation independent, then the bulk of the middling and sub-middling sphalerite in the Cu-circuit concentrates must be locked with chalcopyrite. Thus, regrinding the Cu-scavenger concentrate in which these fractions will tend to concentrate, should lower Cu losses to the Zn-circuit, as well as Zn losses to the Cu-circuit.

Free and sub-free sphalerite comprises 91.5 % of the Zn in the feed, and this constitutes excellent 'effective liberation', as all these grains should behave as free grains of sphalerite during flotation (See Fig. 12). Of this 10.3 % reports to the Cu-concentrate, such that 9.40 % of the Zn in the feed is recovered in the Cu-concentrate as free and sub-free classes of sphalerite. Thus, this principal fraction of the sphalerite that reports to the Cu-concentrate, does not exhibit poor liberation, and therefore indicates poor selective flotation of sphalerite. These recoverable classes of sphalerite, which comprise the bulk (86 %) of the sphalerite in the Cu-concentrate, are exhibiting flotation behaviour that appears to be independent of their state of liberation, and is therefore not expected to be amenable to regrinding as a corrective measure.

The bulk of the recoverable classes of sphalerite (86.7 %) is recovered in the Zn-concentrate, and constitutes a Zn recovery of 79.4 % in the form of free and sub-free sphalerite. Much of the sphalerite classified as sub-free consists of: (1) chalcopyrite-disease in sphalerite; (2) sphalerite-chalcopyrite particles of granular and diablastic character; and (3) sphalerite-pyrite ± arsenopyrite particles. The first two varieties contribute

TABLE 34

MODAL FRACTION OF THE Zn RECOVERY OCCURRING IN COMPOSITE CLASSES OF UNLIBERATED SPHALERITE IN THE 17 MILL PRODUCTS.

PRODUCT.	Zn RECOVERY	ADJUSTED % FREE	ADJUSTED % LOCKED	LOCKED 0-99.9	COMPOSITE CLASSES OF UNLIBERATED GRAINS. 0-30	30-70	0-70	70-99.9	70-100	FREE 99.9-100
1. Feed	100.0	50.5	46.1	46.1	1.86	4.56	8.40	37.6	91.5	50.5
2. Cu-Scav Co.	8.32	14.4	87.4	7.27	0.37	0.71	1.09	6.16	7.22	1.20
3. Cu-Roug Co.	26.8	37.6	61.6	16.5	0.96	2.25	3.41	12.6	22.9	10.1
4. Cu-Cl.1 Fd.	30.0	40.0	58.8	17.6	1.00	2.46	3.67	13.5	25.8	12.1
5. Cu-Cl.1 Co.	14.2	47.2	50.8	7.19	0.46	1.10	1.71	5.41	12.3	6.68
6. Cu-Cl.2 Co.	11.0	42.8	55.4	6.08	0.42	0.89	1.45	4.56	9.40	4.70
7. Zn-Roug Fd.	90.7	50.9	45.6	40.5	2.36	3.88	8.05	33.2	82.7	46.2
8. Zn-Roug Co.	84.2	54.9	41.1	34.6	1.64	3.18	4.63	30.4	79.9	46.2
9. Zn-Cl.1 Co.	87.6	53.1	43.0	37.7	3.45	4.49	6.92	30.4	80.3	46.5
10. Zn-Cl.2 Co.	82.5	55.6	40.3	33.2	0.72	2.97	3.53	30.2	79.4	45.9
11. Cu-Roug Tl.	97.3	48.3	48.6	47.3	1.81	4.38	8.04	39.2	89.3	47.0
12. Cu-Scav Tl.	89.0	51.5	45.0	40.1	1.44	3.67	6.94	33.0	82.1	45.9
13. Cu-Cl.1 Tl.	15.9	33.9	66.0	10.5	0.54	1.36	1.96	8.04	13.5	5.37
14. Cu-Cl.2 Tl.	3.18	62.2	35.0	1.11	0.04	0.21	0.26	0.85	2.91	1.98
15. Zn-Cl.1 Tl.	1.70	21.7	78.2	1.33	0.92	0.21	1.10	0.21	0.57	0.37
16. Zn-Cl.2 Tl.	5.08	12.3	86.2	4.38	1.87	1.52	3.38	0.28	0.91	0.63
17. Zn-Roug Tl.	6.54	0.05	104.3	6.82	0.72	0.71	3.41	2.83	2.75	0.00

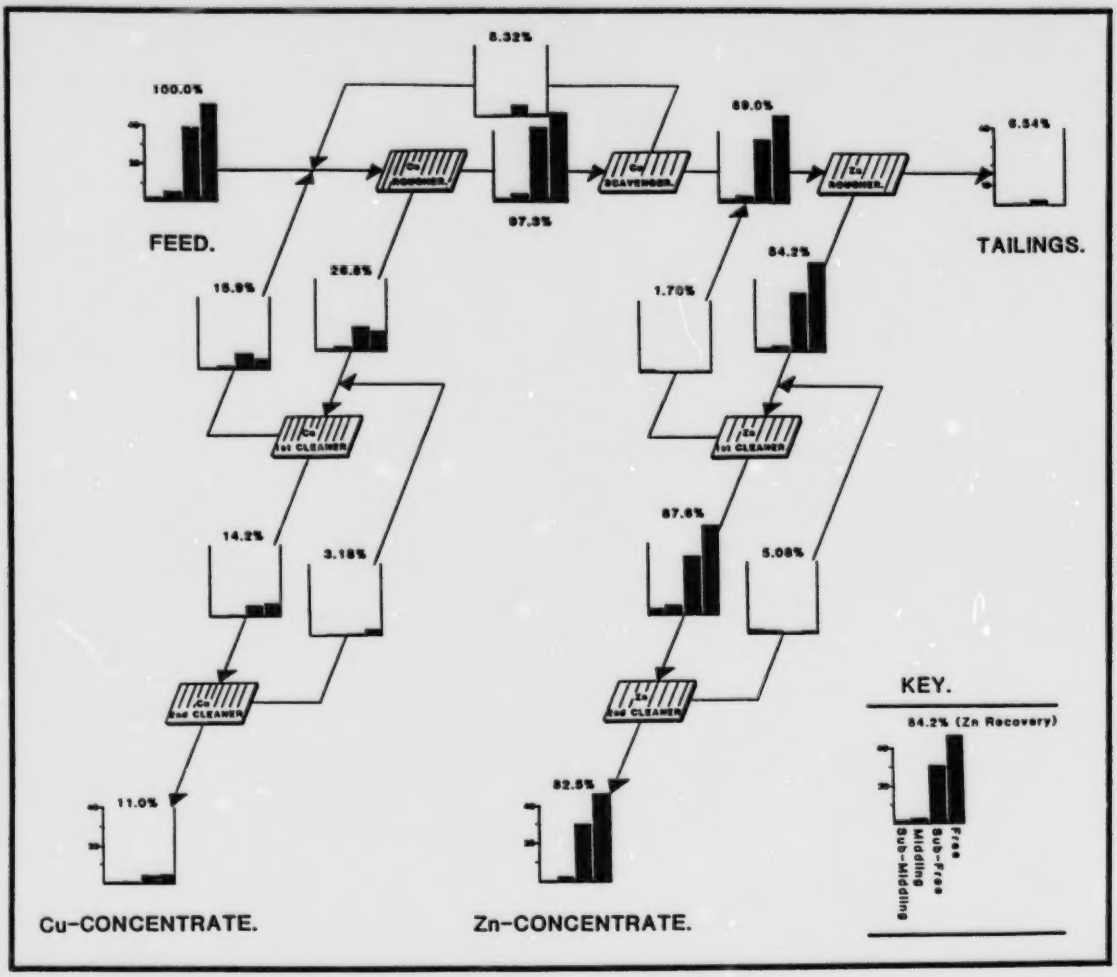


FIGURE 12.
THE Zn RECOVERY MODELLED THROUGH THE CIRCUIT IN TERMS OF THE MODAL FRACTIONS OCCURRING AS FREE, SUB-FREE, MIDDLING AND SUB-MIDDLING SPHALERITE GRAINS. CORRESPONDING DATA GIVEN IN TABLE 34.

to the small (1.56 %) Cu lossses to the Zn-concentrate attributed to sub-middling chalcopyrite, and are therefore essentially unrecoverable to the Cu-concentrate. The Cu-scavenger tailings recovers 82.1 % of the Zn in the feed that occurs in the form of free and sub-free sphalerite, and as stated earlier 96.7 % of this reports to the Zn-concentrate, and constitutes 79.4 % of the Zn recovery. An additional 3.53 % of the Zn in the feed is recovered in the Zn-concentrate as middling and sub-middling sphalerite. Thus, of the 82.5 % Zn recovery in the Zn-concentrate, 96 % consists of free and sub-free sphalerite. This contrasts with the 11.0 % Zn recovery in the Cu-concentrate, of which 86 % consists of free and sub-free sphalerite. Thus, there is a proportionally larger component of middling and sub-middling sphalerite in the Cu-concentrate than in the Zn-concentrate (See Figs. 12 and 33).

The cross-balancing of the material balances in the tailings indicates that 2.75 % of the Zn in the feed is recovered as free plus sub-free sphalerite, whereas 2.83 % of the Zn in the feed is recovered as just sub-free sphalerite. This discrepancy results from the lack of free sphalerite in the tailings and the small and acceptable deviation of the material balances of the separate free and free plus sub-free composite classes. Thus, essentially 2.75 % of the Zn in the feed is recovered in the tailings as free and sub-free sphalerite. This constitutes an overall 3.01 % loss of these specific classes of sphalerite, and a loss of 3.4 % of the recoverable sphalerite entering the Zn-circuit. Thus, the Zn-circuit is achieving greater than 96 % efficiency in the recovery of the free and sub-free composite classes of sphalerite. Nonetheless, and similarly to chalcopyrite, because the free and sub-free fractions comprise 91.5 % of the sphalerite in the feed, these minimal losses of 2.75 % of the Zn to the tailings as free and sub-free sphalerite constitute 42 % of the 6.54 % Zn lost to the tailings. The combined middling and sub-middling composite class comprises an additional 52 % of the Zn lost to the tailings, while the remaining 6 % was unaccounted for on recombining the material balances. Despite the poor cross-balancing for the middling and sub-middling composite classes in the tailings (i.e. $0.72 + 0.72 = 1.43 < 3.41$), and considering that sub-middling particles are more likely to be lost to the tailings, and the dominance of the middling (4.56 %) over the sub-middling composite class (1.86 %) in the feed, it is interpreted that middling and sub-middling composite classes each contribute essentially comparable losses of approximately 1.6 % of the Zn in the feed to the tailings.

The principal fractions of sphalerite lost to the tailings and Cu-concentrate, and whose recovery in the Zn-concentrate would variably contribute to improved Zn-recovery are: (1) the 0.89 % to 1 % (interpreted) Zn loss to the Cu-concentrate as middling sphalerite, which is amenable to regrinding; (2) the 0.71 % to 1.6 % (interpreted) Zn loss to the tailings as middling sphalerite, which is amenable to regrinding; (3) the 2.8 % Zn loss as free and sub-free sphalerite to the tailings, which is nominally not amenable to regrinding, but might be partly recovered through improved flotation efficiency in the Zn-circuit, if the latter can be expected to exceed 97 %; and (4) the 9.40 % Zn loss to the Cu-concentrate as free and sub-free sphalerite, which is also nominally not amenable to regrinding. It is this latter fraction, the principal fraction of the lost sphalerite, whose mineralogical character, mechanism of premature flotation (i.e. prior to activation with $CuSO_4$), and resolution is the principal object of investigation in this study.

The raw % unliberated sphalerite in various selected categoriess of particles are presented in Table 35. The data in the columns labelled 'Cpy', 'Py+Po+Asp' and 'Sulf' represent subsets of that in 'All', where the latter represents the modal percent of sphalerite occurring in particles containing any other mineral, and is therefore equivalent to the % unliberated sphalerite. Combining the values in 'Sphal' and 'Py+Po+Asp' for a given product consistently yields totals in excess of that in 'Sulf' and 'All'. This disparity results from the unliberated sphalerite occurring in particles that may contain grains of one or more other minerals, and thus be identified as occurring in several of the selected categories.

In the feed, 41.8 % of the sphalerite is free, while 58.2 % is unliberated (according to the raw data) with 66 % (i.e. 38.6 %) of this sphalerite locked with sulfides, 62 % (i.e. 35.8 %) is locked with pyrite, pyrrhotite and/or arsenopyrite, and 36 % (i.e. 21.2 %) is locked with chalcopyrite. Comparing the values for 'Sulf' and 'All', at least 20 % of the unliberated sphalerite is also locked with gangue minerals. Thus, pyrite, pyrrhotite and/or arsenopyrite constitute the predominant accompanying mineral(s) in particles containing unliberated sphalerite, as was predicted by Healy & Petruk (1988). This results from sphalerite enveloping, invading and replacing the disseminated pyrite and arsenopyrite in Massive Sphalerite and Banded Pyrite + Sphalerite ore types, and occurring interstially to polygonal pyrite in the Massive Pyrite ore type. While only 8 % (i.e. 3.9 % out of 50.3) of the unliberated sphalerite in the Cu-concentrate is locked with pyrite, pyrrhotite and/or arsenopyrite, 52 % (i.e. 19.2 % out of 36.8) of the unliberated sphalerite in the Zn-concentrate is locked with pyrite, pyrrhotite and/or arsenopyrite. The values for the tailings were erased during data processing, but based on the Zn-1st and 2nd cleaner tailings, in which 70 % and 82 %, respectively of the unliberated sphalerite is locked with pyrite, pyrrhotite and/or arsenopyrite, it is probable that at least 80 % of the unliberated, and hence total sphalerite in the tailings is locked with pyrite, pyrrhotite and/or arsenopyrite.

With the exception of the Zn-2nd cleaner tailings and the Zn-rougher tailings, the Cu-scavenger concentrate contains the highest % unliberated sphalerite i.e. 85.6 %. Even more significant is that 94 % (i.e. 80.2 % of 85.6) of this unliberated sphalerite is locked with chalcopyrite. The magnitude of this value suggests that not only middling and sub-middling sphalerite locked with chalcopyrite is floated. However, sub-free sphalerite, which comprises 74 % of the sphalerite in the Cu-scavenger concentrate, is also floated due to the small volume % (i.e. less than 30 %) of accompanying chalcopyrite grain(s) in the host particle. This suggests that the sphalerite is only sub-marginally floatable, and that if chalcopyrite forms only a relatively small surface area of the particle, it can nonetheless provide sufficient collection and flotation, not typically expected on the basis of the particle classification (See Fig. 13).

Also significant is the progressively increasing % free sphalerite (i.e. from 14.4 % to 49.7 %) passing from the Cu-scavenger, Cu-rougher, Cu-1st cleaner to Cu-2nd cleaner concentrates (See Fig. 12), and the progressive fall in the percentage of the unliberated sphalerite locked with pyrite, pyrrhotite and/or arsenopyrite. The percent of the unliberated sphalerite locked with chalcopyrite is 94 % in the Cu-scavenger concentrate, 80 % in the Cu-rougher

concentrate, 89 % in the Cu-1st cleaner concentrate, and 88 % in the Cu-2nd cleaner concentrate. A large fraction of the unliberated sphalerite locked with chalcopyrite consists of sphalerite that falls in the 60-70 and 70-80 classes, and are medium-grained (e.g. 75 % - 52 um), particularly in the Cu-scavenger concentrate (See Fig. 13). It is noteworthy that the grain size of unliberated sphalerite locked with chalcopyrite becomes progressively finer passing from the Cu-scavenger concentrate (See Fig. 13). While a minor fraction of the sphalerite grains in the 90-99.9 class, and which are locked with chalcopyrite, may be artifact-derived (See Petruk 1987), being actually free sphalerite grains, most of the unliberated sphalerite is actually locked with chalcopyrite. This is supported by petrographic examination of these products.

Regrinding of the Cu-scavenger concentrate, in which 80.2 % of the sphalerite is locked with chalcopyrite, may liberate middling chalcopyrite and sphalerite. It might also improve the selective flotation of the sub-free sphalerite, by liberating the accompanying locked grains of chalcopyrite, which appear to be disproportionately (with respect to volume % of the host particle) influencing the flotation characteristics of sub-free sphalerite. Similarly, the Cu-1st cleaner tailings contains 63.4 % unliberated sphalerite, of which 88 % (i.e. 55.8 %) is locked with chalcopyrite. However, as with the Cu-scavenger concentrate much of this fraction of sphalerite falls in the 90-99.9 class, in which the chalcopyrite may be partly artifact-derived, but most likely principally reflects chalcopyrite-disease as well as granular and diablastic chalcopyrite in sphalerite. The blebs and rods of chalcopyrite in sphalerite due to chalcopyrite-disease are known to be preferentially liberated during grinding, as the sphalerite tends to rupture along the twin planes and fractures, in which the chalcopyrite-disease is concentrated (Petruk & Mainwaring 1985). Thus, provided the sphalerite occurring in the 90-99.9 class principally reflects chalcopyrite-disease, granular or diablastic chalcopyrite, and that the chalcopyrite is exerting a disproportionate influence on the flotation of sphalerite, regrinding of the combined Cu-scavenger concentrate and Cu-1st cleaner tailings may lower the losses of sub-free sphalerite (in addition to that of the smaller component of middling sphalerite) to the Cu-concentrate.

As stated earlier, presently only 60 % (split stream) of the Cu-rougher tailings is reground, and constitutes a stream flow rate of approximately 57. This reduced stream contains 50 to 65 % of each of the four principal composite classes (i.e. free, sub-free, middling and sub-middling) of sphalerite, of which approximately 50 % is locked with chalcopyrite. In contrast, the proposed combined Cu-scavenger concentrate and Cu-1st cleaner tailings would contain approximately 10 % to 25 % of each of these four composite classes of sphalerite, of which approximately 90 % is locked with chalcopyrite, but in a greatly reduced stream (i.e. flow rate = 7.60).

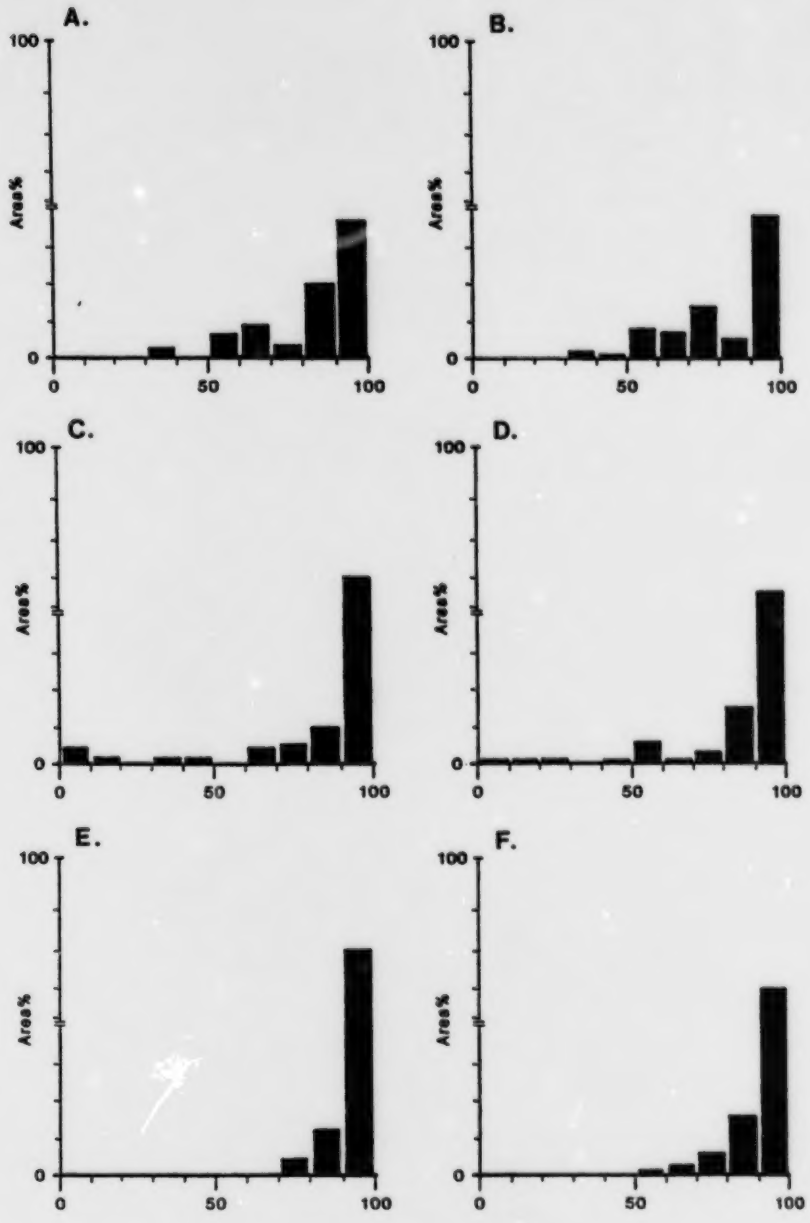


FIGURE 13.
PLOT OF THE SPHALERITE Vs CHALCOPYRITE CLASSIFICATION IN THE (A) FLOAT FEED; (B) Cu-ROUGHER CONCENTRATE; (C) Cu-1ST CLEANER CONCENTRATE; (D) Cu-2ND CLEANER CONCENTRATE; (E) Cu-SCAVENGER CONCENTRATE; AND (F) Cu-1ST CLEANER TAILINGS.

TABLE 35

CONDENSED RAW LIBERATION DATA FOR SELECTED CATEGORIES OF UNLIBERATED SPHALERITE IN THE 17 MILL PRODUCTS.

(in Area % sphalerite in each category).

PRODUCT.	FREE GRAINS (% Free)	UNLIBERATED GRAINS			
		All.	Cpy.	Py+Po+Asp.	Sulf.
1. Feed.	41.8	58.2	21.2	35.8	38.6
2. Cu-Scav Co.	14.4	85.6	80.2	22.9	82.3
3. Cu-Roug Co.	39.7	60.3	48.0	22.1	58.6
4. Cu-Cl.1 Fd.	34.0	66.0	62.5	11.3	63.8
5. Cu-Cl.1 Co.	44.9	55.1	48.8	8.2	49.7
6. Cu.Cl.2 Co.	49.7	50.3	44.2	3.9	44.2
7. Zn-Roug Fd.	57.4	42.6	22.0	30.8	35.2
8. Zn-Roug Co.	58.7	41.3	22.8	12.2	32.7
9. Zn-Cl.1 Co.	61.8	38.2	20.4	12.0	31.7
10. Zn-Cl.2 Co.	63.2	36.8	13.9	19.2	32.9
11. Cu-Roug Tl.	43.0	57.0	28.5	34.4	46.2
12. Cu-Scav Tl.	50.4	49.6	39.7	37.4	46.5
13. Cu-Cl.1 Tl.	36.6	63.4	55.8	12.7	59.7
14. Cu-Cl.2 Tl.	65.6	34.4	29.0	6.0	32.4
15. Zn-Cl.1 Tl.	21.7	78.3	24.4	54.5	70.8
16. Zn-Cl.2 Tl.	12.3	87.7	11.7	71.5	74.9
17. Zn-Roug Tl.	0.0	100.0	----	----	----

Note : '----' denotes data lost during processing.

GALENA

The raw liberation and classification data for galena in each of the 25 mill product samples (compiled from the raw % free and raw galena Vs all classifications) are given in Table 12 of appendix B, and are condensed for the 17 mill products in Table 36. Each of the values in Table 26 were multiplied by the corresponding adjusted Pb assay for that product (See CHEMICAL ANALYSIS) to yield mineral-metal data representing the raw modal fraction of the adjusted Pb assay occurring as (1) unliberated grains; (2) as unliberated grains in each class; and (3) as free grains. This mineral-metal data is given in Table 13 of appendix B, and the data for each of the ten classes of unliberated galena formed the input data for processing by MATBAL. The specific purpose of the later was to generate adjusted mineral-metal data values, from which adjusted modal amounts of unliberated galena in each class could be extracted, and to determine the recovery of each class of unliberated galena in the various products throughout the circuit.

Before proceeding to deal with the results of the material balance calculations of unliberated galena, it is worth examining the raw liberation and classification data for galena in the feed (See Table 36). In terms of the principal composite classes, 8.25 % occurs as free grains, 29.7 % occurs as sub-free grains, 19.15 % occurs as middling grains, and 42.9 % occurs as sub-middling grains of galena. The classification and % free galena in the feed is essentially bimodal with the bulk of the galena occurring in the 0-10 (31.8 %) and in the essentially free (i.e. 90-99.9 and free) composite class (31.5 %). Thus, the counting statistics for galena in the individual classes other than the 0-10 and 90-99.9 classes are poor, not only in the feed, but also particularly in products where these classes are preferentially depleted. Consequently, it is expected that only the 0-10 and 90-99.9 (and possibly also the free, 30-40 and 50-60) classes will yield precise and accurate material balances.

The calculated recovery of the free, of the unliberated, and of each class of the galena classification in each product is presented in Table 37. The data in the columns labelled 'Pb Recovery', '% Locked', and '% Free' have been introduced in previous sections and are presented here for ease of comparison. Seven of the individual classes of unliberated galena gave apparently satisfactory balances, while the remaining three classes each gave one negative recovery value, indicating unsatisfactory balances. It is noteworthy that the 0-10 and 90-99.9 classes both gave good balances and that these classes report principally to the Zn-concentrate and tailings, respectively. The 58.0 % of the 0-10 class of galena that reports to the Zn-concentrate is readily explained in terms of the recovery of composite particles consisting of sphalerite containing fine-grained inclusions of galena, as observed petrographically, and on visual examination of the BSE image of the sample during image analysis. This common occurrence of galena locked with sphalerite originates from the predominant textural association of galena in the ore; galena occurs as blebs in, and as interstitial films and masses along the grain boundaries of, coarse-grained sphalerite (Healy & Petruk 1988). In the case of the 90-99.9 class, the Zn-concentrate and the tailings both recover > 40 % of this class of galena. These grains should behave as free grains of galena, and thus are expected to report to the Cu-concentrate, which recovers only 6.24 % of this class, or to the tailings if galena exhibits inert characteristics during flotation with Minerec.

Stereological biases operating on the small component of the galena that actually occurs as free grains tends to obscure the flotation behaviour of the free galena. Nonetheless, in the section on FREE GRAINS it was interpreted that free galena may report principally to the Cu-concentrate. This was based on the assumption that the observed free grains in the Cu-circuit tailings and Zn-circuit products may be largely stereologically derived from actually unliberated grains. Similarly, it was assumed that the modest increase in the % free, and recovery of free grains of galena through Cu-cleaning may have resulted from observations of apparent free grains of galena in the corresponding tailings, and that in fact free galena grains were recovered in the Cu-concentrate. The low apparent recoveries may have been due to masking by stereologically derived apparent free grains. The magnitude of such a stereological bias in 'natural' (i.e. non-mathematically simulated) systems is typically considerably less than predicted from mathematical simulations (W. Petruk, pers. comm. 1988), but is not measurable for this dataset. Nonetheless, the behaviour of sub-free grains of galena, which are expected to behave as free grains during flotation, and which are not as susceptible to stereological biases (other than shifting into adjacent classes of comparable flotation character anyway), permits a more definitive interpretation of the seemingly conflicting flotation behaviour of free galena. The 90-99.9 class is considered relatively, stereologically representative, and being twice as abundant as the apparent free grains, also contains less uncertainty due to poor counting statistics. Thus, in that only 6.24 % of the 90-99.9 class reports to the Cu-concentrate, with 41.6 % reporting to the Zn-concentrate and 52.2 % to the tailings, it is probable that galena is only marginally flotable (i.e. the Pb-Minerec molecule is probably partly soluble, and forms an unstable layer of surfactant molecules). However, it seems that the galena is rendered more hydrophobic during the additional conditioning period in the Zn-circuit. Thus, the interpretation of the behaviour of essentially free galena during flotation involves (1) possible masking by stereological biases, and/or more probably, (2) non-specific or marginal flotation characteristics of galena.

The adjusted modal fractions of the Pb assay (assumed to be equivalent to the modal abundances of galena) occurring in composite classes of unliberated and free galena are presented in Table 14 of appendix B. These composite classes with relatively uniform and predictable flotation characteristics contain less uncertainty due to counting statistics than the constituent individual classes. The calculated recovery for these composite classes are presented in Table 38. Each of these principal composite classes gave apparently satisfactory material balances with no negative recovery values generated.

The Cu-concentrate recovered 37.1 % of the free and 11.2 % of the unliberated galena, resulting in the overall low recovery of 13.1 % of the Pb. The recovery of 37.1 % of the free galena can be explained by (1) stereologically derived apparently free grains masking the behaviour of free galena, which may indeed be largely recovered in the Cu-concentrate, and/or (2) the flotation properties of galena are marginal, with the Minerec collector forming soluble/unstable hydrophobic films on the galena. As only 11.7 % of the sub-free grains of galena (which are essentially free from bias, and which should behave as free grains of galena) are recovered in the Cu-concentrate, the second interpretation is favoured, or simply predominates.

The highest recovery of any of the principal composite classes of galena into the Cu-concentrate is observed for the sub-middling grains (i.e. 23.9 %). It is interpreted that these grains consist of blebs and fine-grained inclusions in chalcopyrite, most probably derived from Sheared Chalcopyrite + Sphalerite and Chalcopyrite Stringer ore types, in which they are a relatively common association (Healy & Petruk 1988). It is noteworthy, that while the recovery of free, sub-free and middling galena in the Cu-scavenger is 7.23 %, 5.85 % and 1.88 %, respectively, the recovery of sub-middling galena is 17.9 %. This suggests that much of the sub-middling galena occurs in particles consisting of middling or coarse-grained (i.e. 'over-sized'), sub-free chalcopyrite (predominant forms of chalcopyrite likely to be recovered here) that is not recovered initially by the Cu-rougher and hence also contains an inert mineral most probably pyrite. This type of particle consisting of equivalent amounts of pyrite and chalcopyrite and lesser galena, in the form of chalcopyrite and galena enveloping and filling interstices of clusters of pyrite grains, is one of the more common types of unliberated chalcopyrite particles. The Cu-scavenger tailings recovers 62.9 % of the free galena and 88.8 % of the unliberated galena (i.e. 76 % of the middling and of the sub-middling, and 88 % of the sub-free), which hence reports to the Zn-circuit. The Zn-concentrate recovers 41.2 % of the free, 49.5 % of the sub-free, 39.0 % of the middling and 50.1 % of the sub-middling galena. In terms of the amount in each composite class entering the Zn-circuit, the recovery is 65.5 % of the free, 56.1 % of the sub-free, 51.3 % of the middling and 65.7 % of the sub-middling grains. This suggests that the behaviour of galena is relatively independent of the accompanying mineral(s) in a composite particle, anomalously showing moderately good recovery in the Zn-concentrate, irrespective of its state of liberation. It is noteworthy that sub-free and free galena grains exhibit high circulating loads in the Zn-1st cleaner tailings, and extremely high circulating loads in the Zn-2nd cleaner tailings (e.g. 256.4 % recovery of the free grains). As both these composite classes are expected to behave as free grains during flotation, and irrespective of possible stereologically derived apparent free grains (which are statistically likely to be predominantly generated from sub-free grains anyway), this lends further support to the interpretation that galena is marginally floatable.

Of the galena in the feed, 21.7 % of the free and 47.0 % of the unliberated grains are lost to tailings. Healy & Petruk (1988) predicted that galena forming interstitial films and wedges, and dendritic replacements and fracture fillings in pyrite would be poorly liberated and report largely to the tailings. However, as 21.7 % of the free grains and 38.8 % of the sub-free grains are also lost to the tailings, it is probable that galena is at least partly inert, and that pyrite or gangue are not essential accompanying minerals in order to explain large losses of galena to the tailings. During petrographic examination of the tailings several sphalerite-galena particles were observed. However, only very rare grains of unliberated galena occurring principally in particles containing pyrite as the dominant accompanying mineral were observed. The highest losses of galena occur in the middling composite class where 37.1 % of these grains are lost to the tailings, though because middling galena is the most subordinate composite class in the feed, it accounts for a small total loss of galena.

The recovery of the adjusted Pb assay in terms of the adjusted fraction of galena as free grains, unliberated grains, and in the principal composite

TABLE 36

CONDENSED RAW LIBERATION AND CLASSIFICATION DATA FOR GALENA IN THE 17 MILL PRODUCTS.

PRODUCT.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.									% FREE
		0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	91.8	31.8	1.65	9.48	8.50	0.00	8.70	1.95	2.71	3.75	23.2	8.25
2. Cu-Scav Co.	91.4	50.9	1.52	9.29	0.80	0.00	0.00	1.42	0.00	7.67	19.9	8.57
3. Cu-Roug Co.	79.2	16.0	5.86	5.20	2.20	2.02	5.16	7.45	5.18	12.4	17.8	20.8
4. Cu-Cl.1 Fd.	69.0	25.5	2.81	5.18	0.00	10.1	0.00	3.92	1.70	0.00	19.8	31.0
5. Cu-Cl.1 Co.	80.6	17.3	15.0	11.5	7.37	3.64	0.60	0.00	1.71	9.26	14.2	19.4
6. Cu-Cl.2 Co.	81.7	21.8	15.1	4.02	2.38	1.10	8.74	0.74	3.06	13.0	11.7	18.3
7. Zn-Roug Fd.	94.5	7.30	4.13	2.23	0.46	3.75	1.99	2.98	3.41	7.20	61.1	5.45
8. Zn-Roug Co.	72.6	17.9	7.49	2.75	3.74	3.10	2.77	1.50	4.16	3.78	25.4	27.4
9. Zn-Cl.1 Co.	67.7	9.03	6.66	1.49	1.49	0.06	0.75	0.40	6.11	4.18	37.8	32.3
10. Zn-Cl.2 Co.	87.6	13.8	4.69	5.31	0.55	1.99	6.96	1.33	5.07	17.6	30.2	12.4
11. Cu-Roug Tl.	94.3	39.9	14.6	5.97	0.84	3.78	10.6	1.79	0.28	1.00	15.9	5.70
12. Cu-Scav Tl.	93.9	7.78	2.33	1.32	3.03	1.39	1.43	3.20	4.32	8.57	60.6	6.10
13. Cu-Cl.1 Tl.	88.4	11.7	41.3	0.91	1.45	0.72	0.72	3.17	4.53	6.52	17.7	11.6
14. Cu-Cl.2 Tl.	90.4	11.1	9.52	2.83	5.60	7.93	4.94	3.30	2.54	12.5	30.1	9.63
15. Zn-Cl.1 Tl.	65.2	4.06	6.68	0.63	2.09	0.86	0.93	4.95	0.74	6.67	37.6	34.8
16. Zn-Cl.2 Tl.	62.4	0.74	1.48	0.74	1.01	2.78	0.44	1.95	1.88	6.12	45.3	37.6
17. Zn-Roug Tl.	95.1	2.88	9.40	0.18	0.19	0.13	0.48	1.13	2.18	6.21	72.6	4.92

TABLE 37

RECOVERY OF THE FREE, UNLIBERATED AND OF EACH CLASS OF UNLIBERATED GALENA IN THE 17 MILL PRODUCTS.

PRODUCT.	Pb RECOVERY	% LOCKED (0-99.9)	CLASSIFICATION OF UNLIBERATED GRAINS. 0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	% FREE (99.9-100)
1. Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	6.22	6.36	28.3	2.49	14.7	1.80	1.75	1.65	4.03	1.55	10.6	4.83	7.23
3. Cu-Roug Co.	32.0	28.0	44.2	28.5	24.8	16.1	13.6	9.15	36.6	27.6	5.89	21.4	71.1
4. Cu-Cl.1 Fd.	35.2	31.1	47.2	35.7	27.2	20.9	23.2	13.1	40.6	29.9	13.1	25.3	75.2
5. Cu-Cl.1 Co.	16.2	14.4	26.6	37.7	23.1	14.6	17.4	8.19	10.3	7.85	16.2	10.2	41.2
6. Cu-Cl.2 Co.	13.1	11.2	23.6	30.5	20.7	9.68	7.79	4.20	6.40	5.50	8.94	6.24	37.1
7. Zn-Roug Fd.	101.7	99.4	81.6	96.6	81.7	101.3	97.5	100.6	118.7	97.4	117.8	117.5	103.0
8. Zn-Roug Co.	59.9	52.4	63.2	76.7	37.5	20.5	20.5	30.5	35.3	62.6	64.5	65.3	81.3
9. Zn-Cl.1 Co.	91.4	73.0	61.1	67.0	44.0	29.5	7.24	31.2	18.5	88.1	106.4	132.3	297.4
10. Zn-Cl.2 Co.	45.1	41.8	58.0	49.6	35.0	9.57	15.2	25.7	10.1	59.7	37.8	41.6	41.2
11. Cu-Roug Tl.	93.2	95.1	104.6	72.0	93.9	92.1	94.0	97.5	97.6	96.1	101.7	98.6	70.1
12. Cu-Scav Tl.	86.9	88.8	76.4	69.5	79.3	90.3	92.2	95.8	93.6	94.5	91.1	93.8	62.9
13. Cu-Cl.1 Tl.	19.0	16.8	20.6	-2.03	4.09	6.37	5.83	4.96	30.2	20.1	-3.05	15.2	34.0
14. Cu-Cl.2 Tl.	3.18	3.13	3.01	7.15	2.38	4.87	9.60	3.99	3.93	2.35	7.25	3.94	4.12
15. Zn-Cl.1 Tl.	14.8	10.6	5.26	27.1	2.44	11.0	5.30	4.78	25.1	2.93	26.8	23.7	40.1
16. Zn-Cl.2 Tl.	46.3	31.2	3.15	17.4	8.92	19.9	-8.00	5.47	8.40	28.4	68.6	89.8	256.2
17. Zn-Roug Tl.	41.9	47.0	18.4	19.9	44.3	80.7	77.0	70.1	8.40	34.8	53.3	52.2	21.7

classes of unliberated grains in each product is presented in Table 39. These values were generated by multiplying the Pb recovery in a product by the adjusted modal fraction of galena occurring in a specific composite class of unliberated and/or free galena. Thus, the value of 18.5 % under '70-100' grains in the Zn-2nd cleaner concentrate implies that 18.5 % out of the 36.8 % Pb recovery (i.e. 50.3 %) is attributable to free and sub-free or 'recoverable' classes of galena.

Each one of the composite classes of unliberated and/or free galena gave good material balances, without any negative recovery values. Thus, all the columns of data values in Table 39 were extracted from satisfactorily material balanced variables. However, each of the variables was balanced independently of the others, such that the variables do not necessarily cross-balance i.e. sum up to a logical total along each horizontal row, corresponding to a given product. While cross-balancing of independently balanced variables for chalcopyrite and sphalerite were satisfactory, those for galena as shown in Table 39 are extremely poor. In the feed for example, (1) the adjusted % unliberated (90.2 %) and the % free (7.69 %) sum up to 97.9 %; (2) the recovery of Pb attributed to sub-middling (22.3 %), middling (8.22 %), sub-free (26.8 %) and free (7.69 %) galena sum up to 65.0 %; and (3) the recovery of Pb attributed to sub-middling plus middling (31.8 %) and sub-free plus free (37.4 %) galena sums to 69.2 %, all of which should ideally sum up to 100 %. This pattern is repeated in many of the other products, and stems from the modal fraction of Pb occurring in each composite class of galena in the feed being adjusted to a lower value by material balance calculation for each composite class. The reason for the modal fraction of Pb in each composite class of galena in the feed being systematically lowered is not apparent. Possibly, the modal quantities of galena, and hence each composite class of galena, in the feed are excessively high and unrepresentative, due to a sampling or sample preparation bias for the section of the feed i.e. TLMP-1. In consequence, all the modal fractions of the Pb recovery occurring in specific composite classes of galena are depressed, as they are referenced to that in the feed.

While accepting that the data in Table 34 is constrained by very poor cross-balancing, certain conclusions can be drawn form the data. The composition of the adjusted galena classification in the feed is essentially bimodal, with the bulk of the galena occurring in the sub-middling and sub-free composite classes. This pattern is essentially repeated in the Cu- and Zn-concentrates and the tailings, with comparatively minor preferential concentration of a given composite class into a specific end-product. Notably, the sub-middling composite class constitutes the most abundant composite class of the four-fold classification (i.e. 5.22) in the Cu-concentrate, and is marginally less than the sub-free and free composite classes combined (i.e. 6.18). This sub-middling galena occurs as chalcopyrite-galena particles recovered in the Cu-concentrate in response to the flotation characteristics of both chalcopyrite and galena. The abundance of sub-free and free galena, which constitute only 6.18 % of the Pb in the feed, indicates that galena is marginally floatable, though only 16.3 % of the sub-free plus free galena in the feed is recovered in the Cu-concentrate.

The Zn-concentrete recovers the highest fraction of each composite class of galena relative to the Cu-concentrate and tailings. Approximately 15, 5, 13

and 3 % of the Pb in the feed that is recovered in the Zn-concentrate occurs as sub-middling, middling, sub-free and free galena grains, respectively, and constitutes recoveries of between 40 to 50 % for each composite class (See Table 38). This compares with recoveries of between approximately 20 to 30 % for each composite class in the tailings, and which constitute 8.66, 2.81, 9.31 and 1.64 % of the Pb in the feed occurring as sub-middling, middling, sub-free and free galena, respectively. The sub-middling and middling galena recovered in both the Zn-concentrate and tailings consists principally of sphalerite-galena particles, with relatively few galena-pyrite particles, as observed petrographically. The recovery of 13.2 and 9.31 % of the Pb in the feed as sub-free galena in the Zn-concentrate and tailings, respectively is significant. These grains, which are essentially free of stereological biases, and which constitute the most abundant composite class of galena in the feed, probably consist principally of galena-sphalerite and galena-pyrite particles, respectively. Because galena is interpreted to be marginally floatable, being neither inert nor strongly reactive to Minerec, these grains of galena tend to report to a product in accordance with the flotation characteristics of the subordinate (< 30 % of the particle) accompanying unliberated mineral.

Thus, in the Cu-circuit (prior to activation with $CuSO_4$) the apparent order of decreasing floatability is interpreted to be chalcopyrite, galena and sphalerite. In the Cu-concentrate the recovery of free and sub-free grains of (1) chalcopyrite is 99.3 % and 91.2 %, respectively; (2) galena is 37.1 % and 11.7 %, respectively; and (3) sphalerite is 9.29 % and 12.1 %, respectively. The deviation in the behaviour of sub-free galena and sphalerite compared to the free grains of the same minerals, most probably reflects the predominance of chalcopyrite and sphalerite as the principal accompanying unliberated minerals with sub-free sphalerite and galena, respectively, and the influence these exert on the flotation behaviour of the constituent particles.

TABLE 38

RECOVERY OF THE FREE, UNLIBERATED AND OF EACH COMPOSITE CLASS OF UNLIBERATED GALENA IN THE 17 MILL PRODUCTS.

PRODUCT.	Pb RECOVERY	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	90-100	70-100
1. Feed	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	6.22	7.23	6.36	17.9	1.88	13.0	5.85	4.96	5.55
3. Cu-Roug Co.	32.0	71.1	28.0	51.7	40.6	52.3	30.9	31.4	39.3
4. Cu-Cl.1 Fd.	35.2	75.2	31.1	55.2	48.6	56.9	35.9	35.2	44.0
5. Cu-Cl.1 Co.	16.2	41.2	14.4	27.2	31.9	27.4	16.7	16.3	21.0
6. Cu-Cl.2 Co.	13.1	37.1	11.2	23.7	23.9	22.8	11.7	12.5	16.3
7. Zn-Roug Fd.	101.7	103.0	99.4	84.1	96.1	87.5	114.5	122.1	118.6
8. Zn-Roug Co.	59.3	81.3	52.4	57.9	58.9	64.1	75.8	79.5	84.5
9. Zn-Cl.1 Co.	91.4	297.6	73.0	56.6	49.6	67.0	147.6	165.2	170.5
10. Zn-Cl.2 Co.	45.1	41.2	41.8	50.1	39.0	53.9	49.5	44.9	49.5
11. Cu-Roug Tl.	93.2	70.1	95.1	94.3	78.0	90.2	94.1	92.4	89.2
12. Cu-Scav Tl.	86.9	62.9	88.8	76.3	76.1	77.2	88.3	87.5	83.7
13. Cu-Cl.1 Tl.	19.0	34.0	16.8	28.0	16.7	29.5	19.2	18.9	23.0
14. Cu-Cl.2 Tl.	3.18	4.12	3.14	3.47	8.02	4.61	4.95	3.80	4.65
15. Zn-Cl.1 Tl.	14.8	40.1	10.6	7.78	20.0	10.3	26.3	34.7	34.9
16. Zn-Cl.2 Tl.	46.3	256.4	31.2	6.51	10.7	13.2	98.1	120.4	121.0
17. Zn-Roug Tl.	41.9	21.7	47.0	26.2	37.1	23.4	38.8	42.6	34.2

TABLE 39

MODAL FRACTION OF THE Pb RECOVERY OCCURRING IN COMPOSITE CLASSES OF UNLIBERATED GALENA IN THE 17 MILL PRODUCTS.

PRODUCT.	Pb RECOVERY	ADJUSTED % FREE	ADJUSTED % LOCKED	LOCKED 0-99.9	COMPOSITE CLASSES OF UNLIBERATED GRAINS.					FREE 99.9-100
					0-30	30-70	0-70	70-99.9	70-100	
1. Feed	100.0	7.69	90.2	90.2	22.3	8.22	31.8	26.8	37.4	7.69
2. Cu-Scav Co.	6.22	8.93	92.0	5.72	3.96	0.15	4.10	1.58	2.09	0.56
3. Cu-Roug Co.	32.0	17.1	78.6	25.2	11.5	3.31	16.6	8.08	14.8	5.47
4. Cu-Cl.1 Fd.	35.2	16.5	79.6	28.0	12.2	3.96	18.0	9.70	16.5	5.80
5. Cu-Cl.1 Co.	16.2	19.6	79.6	12.9	5.96	2.59	8.63	4.36	7.89	3.18
6. Cu-Cl.2 Co.	13.1	21.9	77.4	10.1	5.22	1.95	7.21	3.18	6.18	2.86
7. Zn-Roug Fd.	101.7	7.90	87.9	89.4	25.1	9.29	31.6	30.5	45.7	8.04
8. Zn-Roug Co.	59.9	10.5	78.7	47.2	16.5	6.47	23.1	20.1	31.5	6.27
9. Zn-Cl.1 Co.	91.4	25.1	71.8	65.7	16.2	6.85	23.8	39.4	62.0	22.9
10. Zn-Cl.2 Co.	45.1	7.04	83.4	37.6	14.7	4.91	19.8	13.2	18.5	3.17
11. Cu-Roug Tl.	93.2	5.66	91.6	85.4	27.4	8.04	32.4	25.2	32.8	5.27
12. Cu-Scav Tl.	86.9	5.67	91.8	79.8	23.4	7.88	28.3	24.8	32.3	4.93
13. Cu-Cl.1 Tl.	19.0	13.8	79.6	15.1	6.28	1.37	9.38	5.13	8.60	2.62
14. Cu-Cl.2 Tl.	3.18	10.0	88.9	2.82	0.77	0.66	1.46	1.32	1.75	0.32
15. Zn-Cl.1 Tl.	14.8	20.9	64.5	9.53	1.76	1.55	3.32	6.99	13.1	3.09
16. Zn-Cl.2 Tl.	46.3	42.6	60.6	28.0	1.31	1.78	3.71	26.2	45.3	19.7
17. Zn-Roug Tl.	41.9	3.91	100.6	42.1	8.66	2.81	8.43	9.31	14.0	1.64

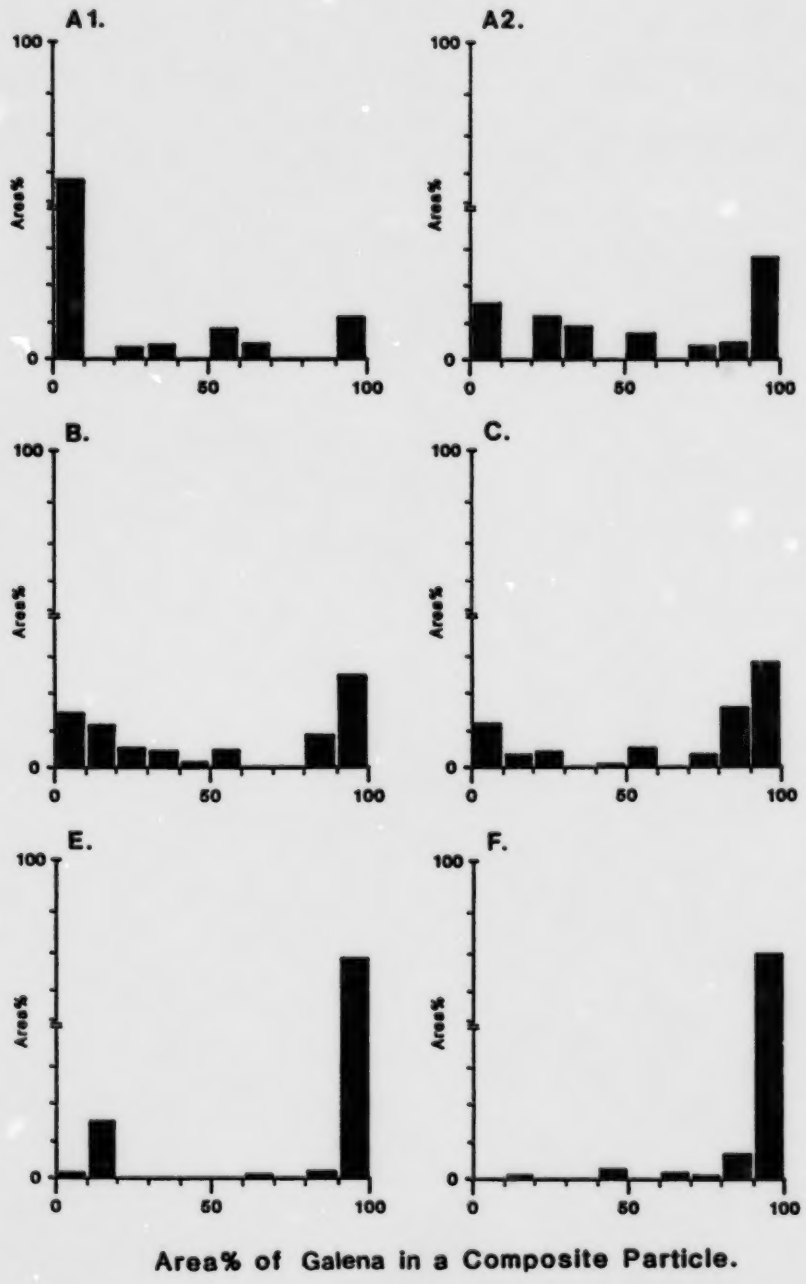


FIGURE 14.
PLOTS OF THE RAW GALENA CLASSIFICATION IN THE (A) FLOAT FEED; (B) Cu-2ND CLEANER CONCENTRATE; (C) Zn-2ND CLEANER CONCENTRATE; (D) Zn-ROUGHER TAILINGS; AND (E) Zn-2ND CLEANER TAILINGS.

PLATE 2.

PLATE 2.

(A) Zn-CONCENTRATE : Conspicuous free, 'over-sized' chalcopyrite, and diablastic sphalerite-chalcopyrite-pyrrhotite particles. Note also middling pyrite-chalcopyrite (A) and pyrite-sphalerite (B). No free pyrite. Plane Polarised Light (PPL).

(B) Zn-CONCENTRATE : Cu losses to the Zn-concentrate are largely as 'over-sized' free grains of chalcopyrite, and as chalcopyrite-disease in, and myrmekitic intergrowths with sphalerite. Note middling chalcopyrite-pyrrhotite particle (A). PPL.

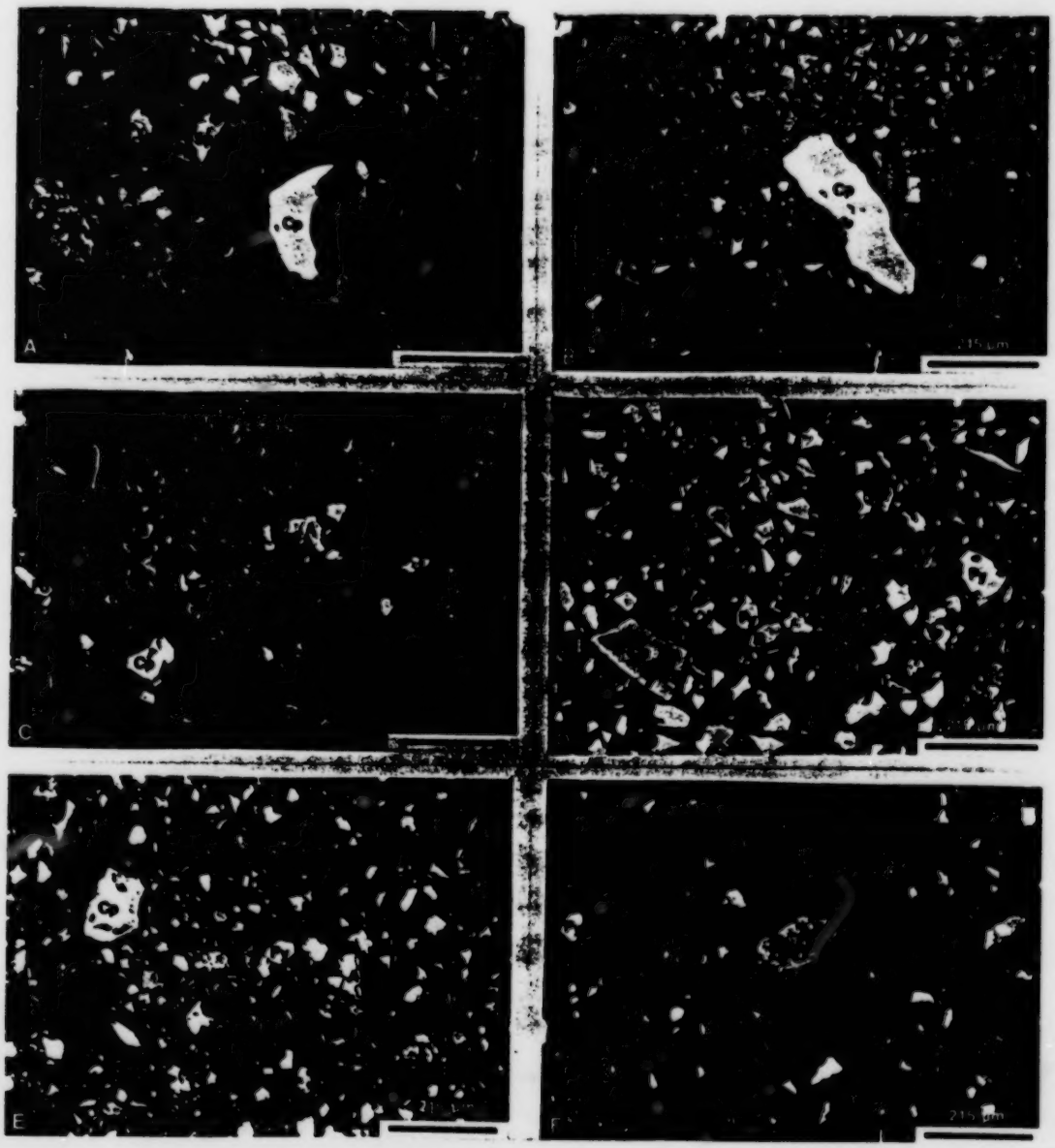
(C) Zn-CONCENTRATE : Free, 'over-sized' chalcopyrite, and abundant sphalerite-chalcopyrite, pyrite-chalcopyrite (A), chalcopyrite-pyrrhotite, pyrite-sphalerite and sphalerite-pyrrhotite middling particles. Not there is negligible free pyrite. PPL.

(D) Cu-SCAVENGER CONCENTRATE : 'Over-sized' chalcopyrite grain and abundant chalcopyrite-sphalerite middling particles consisting of simple granular and diablastic type (A). Bulk of pyrite (C) and pyrrhotite (B) is locked with chalcopyrite. Also note free sphalerite grains. PPL.

(E) Cu-1st CLEANER TAILINGS : Free, 'over-sized' chalcopyrite grain, and abundant chalcopyrite-pyrite (A) and chalcopyrite-sphalerite (B) particles. Note several free grains of sphalerite. PPL.

(F) Zn-ROUGHER TAILINGS : Coarse-grained particle of sphalerite with carious blebs of pyrrhotite (center). Note middling sphalerite-pyrite particle (A). PPL.

PLATE 2.



Chapter VIII

SIZE ANALYSIS

SCREEN SIZE ANALYSIS OF THE ROD MILL FEED

The 10 kg grab sample of the rod mill feed which was collected 'off the belt' midway through the sampling campaign, was split in two. One split was handsorted (See ORE COMPOSOTION OF THE FEED), while the second split was sieved. The handsorted barren waste fraction was also sieved in order to determine whether it is concentrated in the coarser fractions of the feed, and to determine the amount of waste greater than 1 cm in diameter, and hence potentially amenable to photometric ore sorting. Approximately 5 % of the sorted split is less than 3 mm in diameter, and was consequently too small to be visually discriminated, and was rejected. As 55 % of the greater than 3 mm handsorted fraction consisted of waste, it was assumed that approximately half of the less than 3 mm rejected fraction was also waste (i.e. 2.5 %). Thus, the size distribution of the waste fraction was corrected for this fine fraction that was not included in the sieve analysis of the waste (See Table 40).

The results of the screen analysis of the rod mill feed given in Table 40 show that the bulk sample has a size distribution of 80 % - 15 mm, while that of the waste fraction is 80 % - 16 mm. Thus, the waste component of the rod mill feed exhibits a relatively insignificant concentration into the coarser sieve fractions (See Figure 15). Furthermore, 60 % of the waste fraction is greater than 10 mm in diameter, and hence 60 % of the 55 % waste fraction i.e. 33 % of the bulk sample, is in a size range potentially amenable to photometric ore sorting. However, in general photometric ore sorting has achieved very limited success in the pre-concentration of massive sulfide ores. At the Black Angel Mine, central west Greenland, the white marble waste-rock is readily discriminated from ore, and thus facilitated successful application of photometric ore sorting. Perhaps the difference in specific gravity of the silicate gangue minerals (approx. 2.8) and the ore minerals (approx. 4.5) could be exploited through pre-concentration by gravity separation, and thus reduce the large waste fraction.

TABLE 40

SCREEN SIZE ANALYSES OF THE TROUT LAKE ROD MILL FEED (BULK) AND THE HAND-SORTED BARREN WASTE FRACTION.

BULK.			BARREN WASTE FRACTION.			
Screen Opening (mm)	Weight Retained (%)	Passing (%)	Screen Opening (mm)	Weight Retained (%)	Corrected (%)	Passing (%)
38.1	0.00	100.0	38.1	0.00	0.00	100.0
18.9	13.4	86.7	18.9	10.2	9.98	90.4
13.3	47.0	39.7	13.3	51.3	50.0	40.0
6.68	24.9	14.8	6.68	28.2	27.5	12.5
3.33	6.02	8.77	4.70	7.07	6.90	5.64
2.00	3.84	4.93	3.33	2.64	2.58	3.06
-2.00	4.93	----	-3.33	0.63	3.06	----
80 % - 15 mm.			80 % - 16 mm (or 40 % - 10 mm).			

Note : 'Corrected' values account for approximately 2.5 % of bulk occurring as waste, but in the fine fraction (< 3 mm) that was not handsorted.

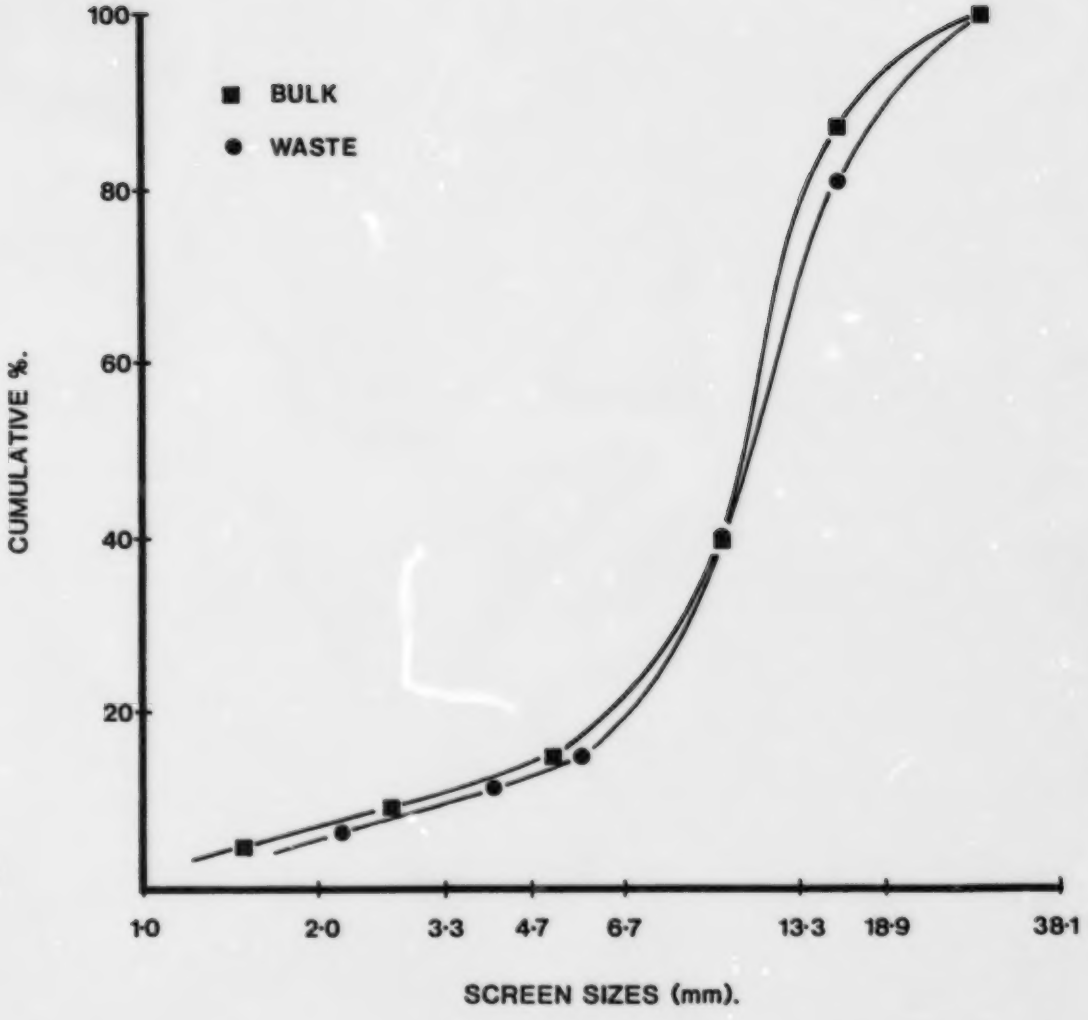


FIGURE 15.
THE CUMULATIVE PARTICLE SIZE CURVES OF THE TROUT LAKE ROD MILL FEED: THE BULK SAMPLE AND THE HAND-SORTED WASTE FRACTION.

SCREEN SIZE ANALYSIS OF THE MILL PRODUCTS

Sieve size analysis using Tyler series meshes 48, 65, 100, 150, 200 and 325 were performed by HBM&S on the 25 mill product samples. The results are presented in Table 41, and are condensed for the 17 principal mill products in Table 42. It can be seen that 67.8 % of the feed is finer than the 325 mesh (i.e. 43 um), and thus lies below the resolution of the technique. Furthermore, as chalcopyrite and sphalerite tend to be concentrated in the finer sieve fractions of the grind, it is likely that an even larger proportion of these minerals of interest are concentrated in the - 325 mesh fraction. In addition, as screen size analysis is a bulk technique it does not address the mineralogical nature of the product, but merely its particulate nature. Nonetheless, screen size analysis does complement the other analytical techniques employed in this study, and highlight certain significant trends.

From the data in Table 42, the raw screen size curves of the four end products (i.e. feed, concentrates and tailings) are plotted in Figure 16A. The Cu- and Zn-concentrates are both significantly finer than the feed suggesting that both chalcopyrite and sphalerite are concentrated in the finer sieve fractions of the feed. However, it may also reflect the differential flotation of finer-grained chalcopyrite and sphalerite, with oversized grains (i.e. > 100 um) being rejected during flotation. In this regard it is noteworthy that both these concentrates contain less than 1.5 % particles greater than 100 um in size, the nominally accepted limit for optimum sulfide flotation.

The particle size curve for the feed is slightly coarser than that of the tailings, despite extraction of the finer sized Cu- and Zn-concentrates. In the absence of a regrind, the extraction of the finer-sized concentrates would concentrate the coarse pyrite, magnetite and silicate minerals (and possibly oversized chalcopyrite and sphalerite), in the tailings, thus generating a slightly coarser sized tailings with respect to the feed. Regrinding 60 % of the Cu-rougher tailings consequently results in the finer sizes of all end products with respect to the feed. However, the tailings, which constitutes 88.5 % of the feed, is only marginally finer than the feed, indicating that the regrind achieved a relatively small increase in size reduction. As chalcopyrite and sphalerite combined comprise only 4.3 volume % of the Cu-rougher tailings, the bulk of the regrind feed consists of inert gangue minerals destined for the tailings, and exhibits an extremely modest fall in particle size. Thus, it appears initially from the raw size data that the regrind is ineffective in generating significant size reduction due to the resistant or 'hard' nature of the product.

The data in Table 42 was transformed by MATBAL into a coherent set of adjusted screen sizes, which are given in Table 43. The recovery of the screen sizes in each product are presented in Table 45. In contrast to all previous material balances so far discussed in this report, the material balancing of these screen analyses employed a 'non-normal' node to account for the change in particle size distribution at the regrind. This facility allows the mass of a given variable to be inequivalent on either side of the node i.e. the mass of coarse particles (in the + 48 and + 65 meshes for example) may be reduced in the regrind tailings. Consequently, the sum of the masses of a given variable (e.g. + 48 mesh fraction) for the three end products differ from that in the feed. Mass conservation is satisfied to the extent that the loss in the

+ 48 mesh fraction is compensated for by increases in the - 325 mesh fraction. However, as MATBAL balances each variable independently (e.g. the + 48 separate from the - 325 mesh fraction), mass conservation is not implicit in the balancing, but is inherent in the raw data on which the balance is performed. The column of data labelled 'Total' refers to the sum of each of the independently balanced screen sizes that comprise the particle size distribution of a given product. As these totals indicate, cross-balancing is not necessarily achieved by MATBAL, though in general the deviation from the ideal total of 100 % is relatively small. However, the deviation in the Cu-circuit concentrates progressively increases through Cu-cleaning e.g. 92.97 in the Cu-2nd cleaner concentrate. The reason for the systematic trend in the discrepancies is not apparent. These derived particle size distributions were normalised to 100 % in order to remove the error in cross-balancing and facilitate plotting (See Table 44).

The normalised adjusted particle size distributions of the feed, concentrates and tailings are plotted in Figure 16B. The curves are generally identical to those of in Figure 16A, indicating that only minor adjustments to the raw data were required by MATBAL to achieve a satisfactory balance. It is apparent from Figures 16A and 16B that the Cu- and Zn-concentrates have generally comparable particle size distributions. It is also apparent from Figure 16B that the adjustments to the raw data forced the curve for the tailings to only a slightly finer particle size distribution than shown in Figure 16A.

The normalised, adjusted particle size distributions of the feed and concentrates of the Cu-circuit and Zn-circuit are shown in Figures 17A and 17B, respectively. The most striking feature of Figure 17A is the coarse size distribution of the Cu-scavenger concentrate. As the Cu-scavenger 'pulls hard' to promote the flotation of marginally floatable categories of minerals, it is expected to recover middling particles of chalcopyrite, oversized free and unliberated chalcopyrite, and other floated and/or entrained mineral species. This is due to the inherently less selective flotation resulting from the elevated setting of the tailings weir height. For a given size grain of chalcopyrite, the particle size equals that of the grain when free, whereas when unliberated, and if occurring in a middling particle, the particle size is much greater than that of the chalcopyrite grain. Thus, middling particles, the recovery of which is the primary objective of a scavenger unit, are more likely to be coarse-grained, whereas free grains tend to be fine-grained.

The second feature of interest in Figure 17A is the increasing coarseness of the particle size distribution of the Cu-rougher concentrate, Cu-1st cleaner concentrate and the Cu-2nd cleaner concentrate. One might expect an opposing trend due to the progressive rejection of middling and oversized chalcopyrite. However, this is clearly countered by the progressive rejection of entrained inert mineral species, which would tend to be finer-grained. At each step in Cu-cleaning, progressively less entrained material is available from the prior step and hence becomes increasingly rejected, thus tending to increase the particle size distribution of the concentrate. This contrasts with the pattern of particle size distributions for the Zn-circuit concentrates in Figure 17B. In these concentrates, there is a dramatic coarsening at the Zn-1st cleaner stage, with the return of a finer tailings to the Zn-rougher feed, thus decreasing its particle size distribution with

TABLE 41

RAW SCREEN SIZE ANALYSES FOR THE 25 MILL PRODUCT SAMPLES.

(Retained Weight i.e. greater than)

SAMPLE.	Mesh Numbers (and Screen Opening Sizes) +48 (295um)	+65 (208um)	+100 (147um)	+150 (104um)	+200 (74um)	+325 (43um)	-325
1. Feed.	0.55	0.77	2.60	5.14	6.74	16.5	67.8
8. Cu-Scav Co.	0.73	1.65	3.39	6.57	12.8	27.2	47.7
2. Cu-Roug Co.	0.02	0.07	0.36	1.38	4.66	16.3	77.2
4. Cu-Roug Co.	0.04	0.06	0.56	1.25	4.55	17.0	76.6
6. Cu-Roug Co.	0.02	0.02	0.25	0.96	2.25	19.0	77.5
15. Cu-Cl.1 Fd.	0.01	0.02	0.23	1.02	2.35	17.7	78.6
10. Cu-Cl.1 Co.	0.18	1.04	2.93	5.38	10.4	21.1	58.9
12. Cu-Cl.2 Co.	0.03	0.04	0.52	1.38	5.27	19.7	73.1
21. Zn-Roug Fd.	0.37	0.83	2.85	5.93	7.79	13.2	69.1
14. Zn-Roug Co.	0.07	0.08	0.68	2.26	4.75	21.5	70.6
16. Zn-Roug Co.	0.07	0.19	0.64	2.15	6.07	16.4	74.5
18. Zn-Roug Co.	0.17	0.19	1.10	2.29	6.50	19.0	70.8
20. Zn-Roug Co.	0.17	0.25	0.85	2.59	5.49	22.3	68.4
22. Zn-Roug Co.	0.14	0.27	0.85	2.57	7.00	17.7	71.5
24. Zn-Cl.1 Co.	0.16	0.31	1.23	3.49	7.12	29.1	58.6
26. Zn-Cl.2 Co.	0.03	0.09	0.35	1.43	5.09	16.1	77.0
3. Cu-Roug Tl.	0.66	1.48	4.29	7.87	9.92	14.2	61.6
7. Cu-Scav Tl.	0.27	0.69	2.48	5.15	7.72	13.0	70.7
9. Cu-Scav Tl.	0.29	0.52	3.64	5.14	8.25	15.4	66.7
11. Cu-Cl.1 Tl.	0.02	0.02	0.18	0.58	2.47	13.0	83.7
25. Cu-Cl.1 Tl.	0.05	0.08	0.31	0.97	1.62	15.5	81.5
13. Cu-Cl.2 Tl.	0.00	0.01	0.12	0.56	2.60	14.7	82.1
23. Zn-Cl.1 Tl.	0.11	0.06	0.42	0.94	1.44	8.14	88.9
19. Zn-Cl.2 Tl.	0.09	0.04	1.08	3.34	11.4	32.5	51.5
17. Zn-Roug Tl.	0.22	0.52	3.09	4.63	7.57	14.5	69.5

TABLE 42

CONDENSED RAW SCREEN SIZE ANALYSES FOR THE 17 MILL PRODUCTS.

(Retained Weight i.e. greater than)

PRODUCT.	Mesh Numbers (and Screen Opening Sizes)						
	+48 (295um)	+65 (208um)	+100 (147um)	+150 (104um)	+200 (74um)	+325 (43um)	-325
1. Feed.	0.55	0.77	2.60	5.14	6.74	16.5	67.8
2. Cu-Scav Co.	0.73	1.65	3.39	6.57	12.8	27.2	47.7
3. Cu-Roug Co.	0.03	0.06	0.46	1.23	4.18	17.2	76.9
4. Cu-Cl.1 Fd.	0.01	0.02	0.23	1.02	2.35	17.7	78.6
5. Cu-Cl.1 Co.	0.18	1.04	2.93	5.38	10.4	21.1	58.9
6. Cu-Cl.2 Co.	0.03	0.04	0.52	1.38	5.27	19.7	73.1
7. Zn-Roug Fd.	0.37	0.83	2.85	5.93	7.79	13.2	69.1
8. Zn-Roug Co.	0.12	0.20	0.82	2.37	5.96	19.4	71.1
9. Zn-Cl.1 Co.	0.16	0.31	1.23	3.49	7.12	29.1	58.6
10. Zn-Cl.2 Co.	0.03	0.09	0.35	1.43	5.09	16.1	77.0
11. Cu-Roug Tl.	0.66	1.48	4.29	7.87	9.92	14.2	61.6
12. Cu-Scav Tl.	0.28	0.61	3.06	5.15	7.98	14.2	68.7
13. Cu-Cl.1 Tl.	0.04	0.05	0.25	0.78	2.05	14.2	82.6
14. Cu-Cl.2 Tl.	0.00	0.01	0.12	0.56	2.60	14.7	82.1
15. Zn-Cl.1 Tl.	0.11	0.06	0.42	0.94	1.44	8.14	88.9
16. Zn-Cl.2 Tl.	0.09	0.04	1.08	3.34	11.4	32.5	51.5
17. Zn-Roug Tl.	0.22	0.52	3.09	4.63	7.57	14.5	69.5

TABLE 43

ADJUSTED SCREEN SIZE ANALYSES FOR THE 17 MILL PRODUCTS.

(Retained Weight i.e. greater than)

PRODUCT.	Mesh Numbers (and Screen Opening Sizes)							Total
	+48 (295um)	+65 (208um)	+100 (147um)	+150 (104um)	+200 (74um)	+325 (43um)	-325	
1. Feed.	0.58	0.91	3.02	5.88	7.68	15.2	64.5	97.8
2. Cu-Scav Co.	0.73	1.66	3.40	6.59	12.9	27.2	47.6	100.0
3. Cu-Roug Co.	0.02	0.03	0.35	1.19	3.52	17.6	74.9	97.6
4. Cu-Cl.1 Fd.	0.02	0.03	0.34	1.15	3.46	17.4	75.3	97.7
5. Cu-Cl.1 Co.	0.02	0.03	0.42	1.44	4.61	19.8	67.3	93.6
6. Cu-Cl.2 Co.	0.02	0.03	0.45	1.54	4.86	20.4	65.7	93.0
7. Zn-Roug Fd.	0.26	0.58	2.92	4.99	7.65	13.9	69.3	99.7
8. Zn-Roug Co.	0.06	0.10	0.43	1.54	4.49	15.8	74.5	97.9
9. Zn-Cl.1 Co.	0.06	0.09	0.70	2.42	7.54	24.2	62.8	97.8
10. Zn-Cl.2 Co.	0.04	0.12	0.43	1.77	5.72	18.8	69.3	96.1
11. Cu-Roug Tl.	0.62	0.99	3.22	6.22	7.99	15.1	64.0	98.2
12. Cu-Scav Tl.	0.26	0.59	2.97	5.06	7.77	14.0	69.0	99.7
13. Cu-Cl.1 Tl.	0.02	0.04	0.22	0.76	1.84	14.0	86.5	103.4
14. Cu-Cl.2 Tl.	0.00	0.01	0.12	0.56	2.55	14.7	80.6	98.5
15. Zn-Cl.1 Tl.	0.13	0.06	0.45	0.98	1.47	8.32	87.4	98.8
16. Zn-Cl.2 Tl.	0.10	0.04	1.22	3.68	11.1	34.6	50.4	101.1
17. Zn-Roug Tl.	0.27	0.62	3.10	5.23	7.87	13.8	69.0	99.9

NOTE : Material balance calculations of the adjusted screen sizes employed a 'Non-Normal Node' to account for the change in grain sizes of the streams due to a 'Regrind' of the Cu-rougher tailings.

TABLE 44

NORMALISED ADJUSTED SCREEN SIZE ANALYSES FOR THE 17 MILL PRODUCTS.

(Retained Weight i.e. greater than)

PRODUCT.	Mesh Numbers (and Screen Opening Sizes)							Normalising Factor.
	+48 (295um)	+65 (208um)	+100 (147um)	+150 (104um)	+200 (74um)	+325 (43um)	-325	
1. Feed.	0.59	0.93	3.09	6.01	7.85	15.6	66.0	1.022
2. Cu-Scav Co.	0.73	1.66	3.40	6.59	12.9	27.2	47.6	1.000
3. Cu-Roug Co.	0.02	0.03	0.36	1.22	3.61	18.0	76.8	1.025
4. Cu-Cl.1 Fd.	0.02	0.03	0.35	1.18	3.54	17.8	77.1	1.024
5. Cu-Cl.1 Co.	0.02	0.03	0.45	1.54	4.93	21.2	71.9	1.069
6. Cu-Cl.2 Co.	0.02	0.03	0.48	1.66	5.23	21.9	70.6	1.076
7. Zn-Roug Fd.	0.26	0.58	2.93	5.01	7.68	14.0	69.6	1.003
8. Zn-Roug Co.	0.06	0.10	0.44	1.59	4.64	16.3	76.9	1.032
9. Zn-Cl.1 Co.	0.06	0.09	0.72	2.47	7.71	24.7	64.2	1.022
10. Zn-Cl.2 Co.	0.04	0.13	0.45	1.84	5.95	19.5	72.1	1.040
11. Cu-Roug Tl.	0.63	1.01	3.28	6.33	8.14	15.4	65.3	1.018
12. Cu-Scav Tl.	0.26	0.59	2.98	5.08	7.80	14.1	69.2	1.003
13. Cu-Cl.1 Tl.	0.02	0.04	0.21	0.74	1.78	13.6	83.7	0.967
14. Cu-Cl.2 Tl.	0.00	0.01	0.12	0.57	2.59	15.0	81.8	1.015
15. Zn-Cl.1 Tl.	0.13	0.06	0.46	0.99	1.49	8.42	88.5	1.013
16. Zn-Cl.2 Tl.	0.10	0.04	1.21	3.64	10.9	34.3	49.8	0.989
17. Zn-Roug Tl.	0.27	0.62	3.11	5.24	7.88	13.8	69.1	1.002

NOTE : Adjusted screen sizes were normalised to 100 %, to account for the lack of cross-balancing of the different sizes in each product during the material balance calculations.

respect to the Cu-scavenger tailings. In the absence of a Zn-scavenger unit, the Zn-rougher tends to 'pull hard' and recover a high proportion of entrained inert fines, which are subsequently rejected by the more selective Zn-1st cleaner. At the Zn-2nd cleaner the rejected particles are preferentially coarser than those of the final Zn-concentrate. With much of the finer entrained inert particles rejected at the 1st cleaner, the rejected material at the 2nd cleaner likely corresponds to middling and sub-middling sphalerite grains that occur in coarse particles. It must be stressed that the use of sieve size analysis to evaluate individual mineral behaviour is conjectural in that (1) it does not address the mineralogical nature, but merely the particulate nature of the bulk product, and (2) as approximately 70 % of the material being examined lies below the resolution of the method, the examination is restricted to the coarse tail of the particle size distribution.

In Figure 18A, the particle size distributions of the potentially regrindable products are compared. Both the Cu-1st and 2nd cleaner tailings have extremely fine particle size distributions compared to the Cu-rougher tailings and particularly the Cu-scavenger concentrate. For reasons of mineralogical composition outined in MINERAL LIBERATIONS, the Cu-scavenger concentrate and the Cu-1st cleaner tailings are preferred streams for regrinding. The extremely coarse nature of the Cu-scavenger concentrate therefore further supports its selection for regrinding, while that of the Cu-1st cleaner tailings, being extremely fine is a less attractive, yet regrindable stream. The particle size distribution of the Cu-rougher tailings lies somewhat midway between these two streams, but for reasons of high stream flow rates and mineralogical composition, namely its pyrite- and gangue-rich nature, is unsuitable for regrinding. In Figure 18B, the particle size distributions of the Cu-rougher tailings (regrind feed) and the combined Cu-scavenger concentrate and tailings (regrind tailings - reconstituted) are compared. At 43 um (i.e. 325 mesh) the regrinding has resulted in a 3 % decrease in particle sizes, or at 80 % passing there is less than a mesh size reduction in the particle size distribution of the regrind tailings. The size distribution of the regrind feed is defined by 80% - 58 um, whereas that of the regrind tailings is defined by 80 % - 50 um, which is only slightly finer than that of the float feed, which is defined by 80 % - 55 um. Such a size reduction is not expected to yield significant liberation of unliberated chalcopyrite or sphalerite, though it may provide for cleaning or 'freshening up' of the mineral grain surfaces.

Th recoveries of the adjusted screen size fractions calculated by MATBAL in each of the products are presented in Table 45. These data are plotted in Figure 19 as histograms portraying the recovery of each screen fraction in each of the 17 mill products. It is apparent that with the exception of the Cu-scavenger concentrate, the Cu- and Zn-circuit concentrates do not recover significant amounts (i.e. > 3 %) except in the + 74, + 43 and - 43 um fractions. This reflects either (1) the concentration of chalcopyrite and sphalerite in the finer size fractions of the grind, or (2) the inability to float oversized chalcopyrite and sphalerite. The Cu-scavenger concentrate, which has a stream flow rate of 1.93 recovers only 1.42 % of the - 43 um fraction, again highlighting the coarse particle size distribution of this stream. The histograms also highlight the progressive loss of the more coarse fractions, due to regrindig. Thus, while the Cu- and Zn-concentrates recover

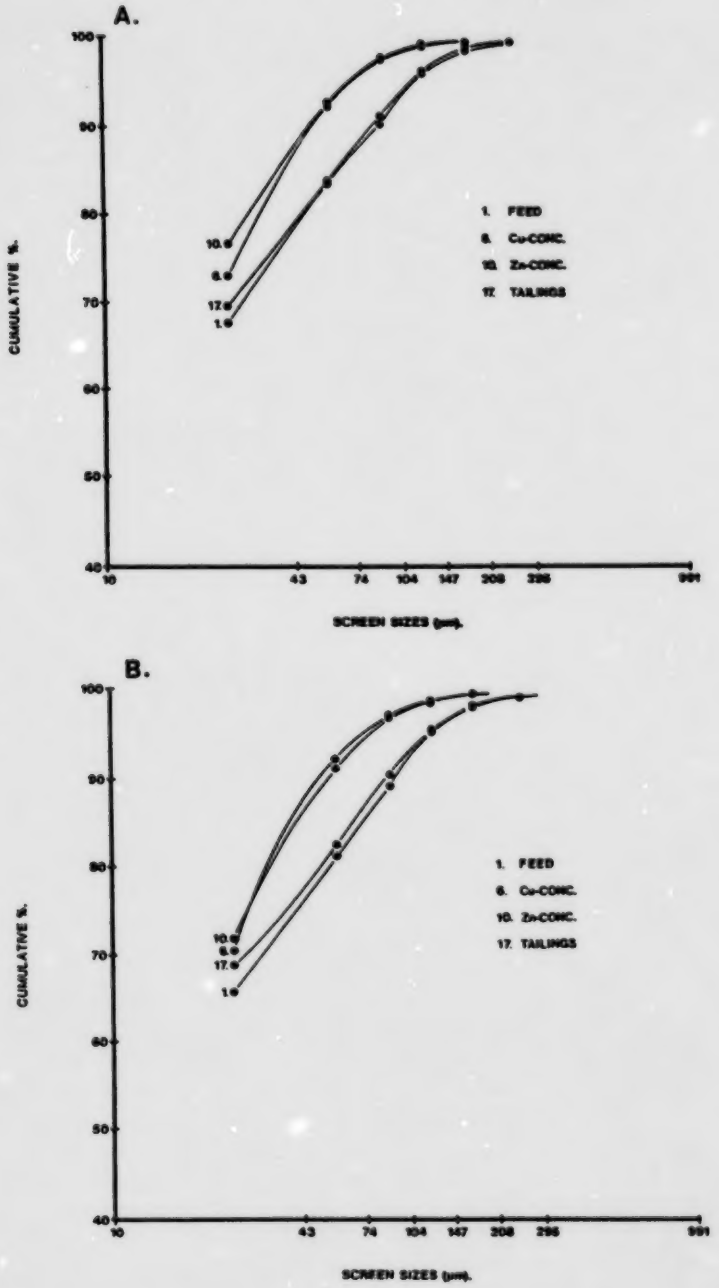


FIGURE 16.
THE PARTICLE SIZE CURVES FOR THE PRINCIPAL END PRODUCTS : (A) RAW, AND (B) NORMALISED, ADJUSTED DATA.

0.22 and 0.27 %, respectively of the + 295 um fraction, only 41.4 % is recovered in the tailings, implying that 58.4 % of the + 295 um fraction was reduced to a finer size fraction by regrinding. Similarly, 39.1 %, 7.43 %, 18.1 %, 1.48 % and 4.85 % were removed from the + 208, + 147, + 104, + 74 um and + 43 um fractions, respectively, and contributed to the 6.56 % increase in the volumetrically predominant - 43 um fraction.

TABLE 45

RECOVERY OF THE SCREENED FRACTIONS FOR THE 17 MILL PRODUCTS.

PRODUCT.	Mesh Numbers (and Screen Opening Sizes) +48 (295um)	+65 (208um)	+100 (147um)	+150 (104um)	+200 (74um)	+325 (43um)	-325
1. Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2. Cu-Scav Co.	2.44	3.52	2.17	2.16	3.23	3.43	1.42
3. Cu-Roug Co.	0.38	0.46	1.48	2.58	5.83	14.7	14.8
4. Cu-Cl.1 Fd.	0.38	0.47	1.51	2.67	6.11	15.5	15.9
5. Cu-Cl.1 Co.	0.22	0.25	1.09	1.93	4.75	10.3	8.24
6. Cu-Cl.2 Co.	0.22	0.24	1.05	1.85	4.47	9.47	7.19
7. Zn-Roug Fd.	42.0	60.8	91.8	80.3	94.4	86.7	101.8
8. Zn-Roug Co.	0.66	0.70	0.91	1.65	3.68	6.52	7.28
9. Zn-Cl.1 Co.	0.68	0.68	1.58	2.80	6.69	10.8	6.64
10. Zn-Cl.2 Co.	0.27	0.58	0.64	1.35	3.34	5.53	4.82
11. Cu-Roug Tl.	102.2	103.3	101.1	100.3	98.8	94.0	94.2
12. Cu-Scav Tl.	41.6	60.7	91.5	80.0	94.1	85.7	99.4
13. Cu-Cl.1 Tl.	0.16	0.23	0.42	0.74	1.36	5.23	7.61
14. Cu-Cl.2 Tl.	0.00	0.01	0.03	0.08	0.28	0.82	1.06
15. Zn-Cl.1 Tl.	0.39	0.13	0.27	0.30	0.35	1.00	2.46
16. Zn-Cl.2 Tl.	0.41	0.11	0.94	1.46	3.36	5.30	1.82
17. Zn-Roug Tl.	41.4	60.1	90.9	78.7	90.7	80.2	94.6

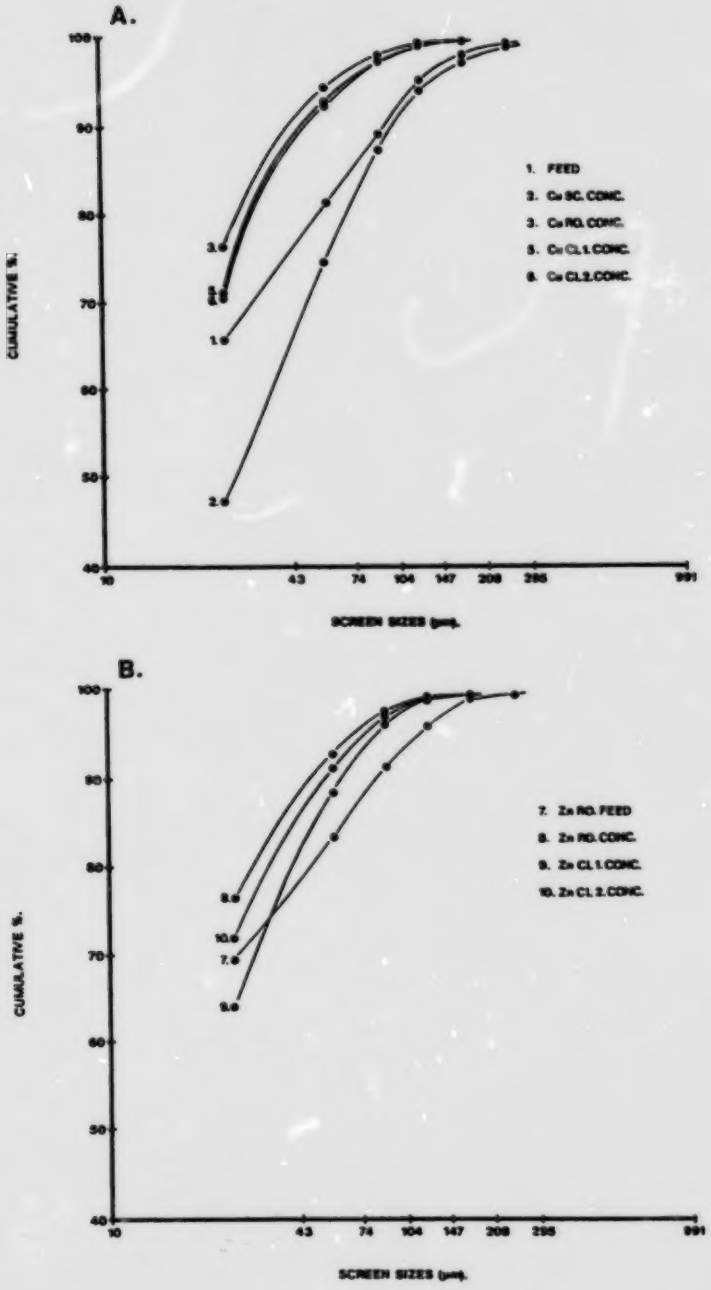


FIGURE 17.
NORMALISED, ADJUSTED PARTICLE SIZE CURVES FOR FEEDS TO, AND CONCENTRATES FROM, (A) THE Cu-CIRCUIT, AND (B) THE Zn-CIRCUIT.

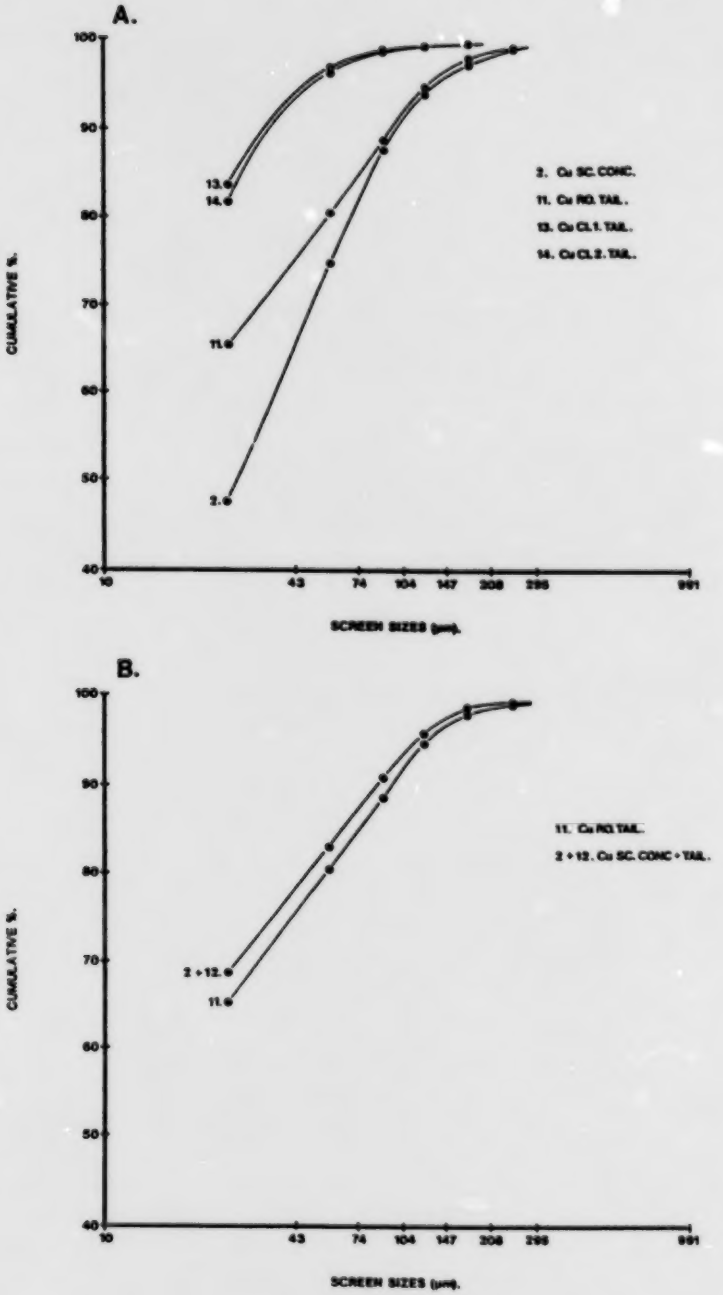


FIGURE 18.
NORMALISED, ADJUSTED PARTICLE SIZE CURVES FOR (A) THE POTENTIALLY REGRINDABLE PRODUCTS, AND (B) THE REGRIND FEED AND TAILINGS.

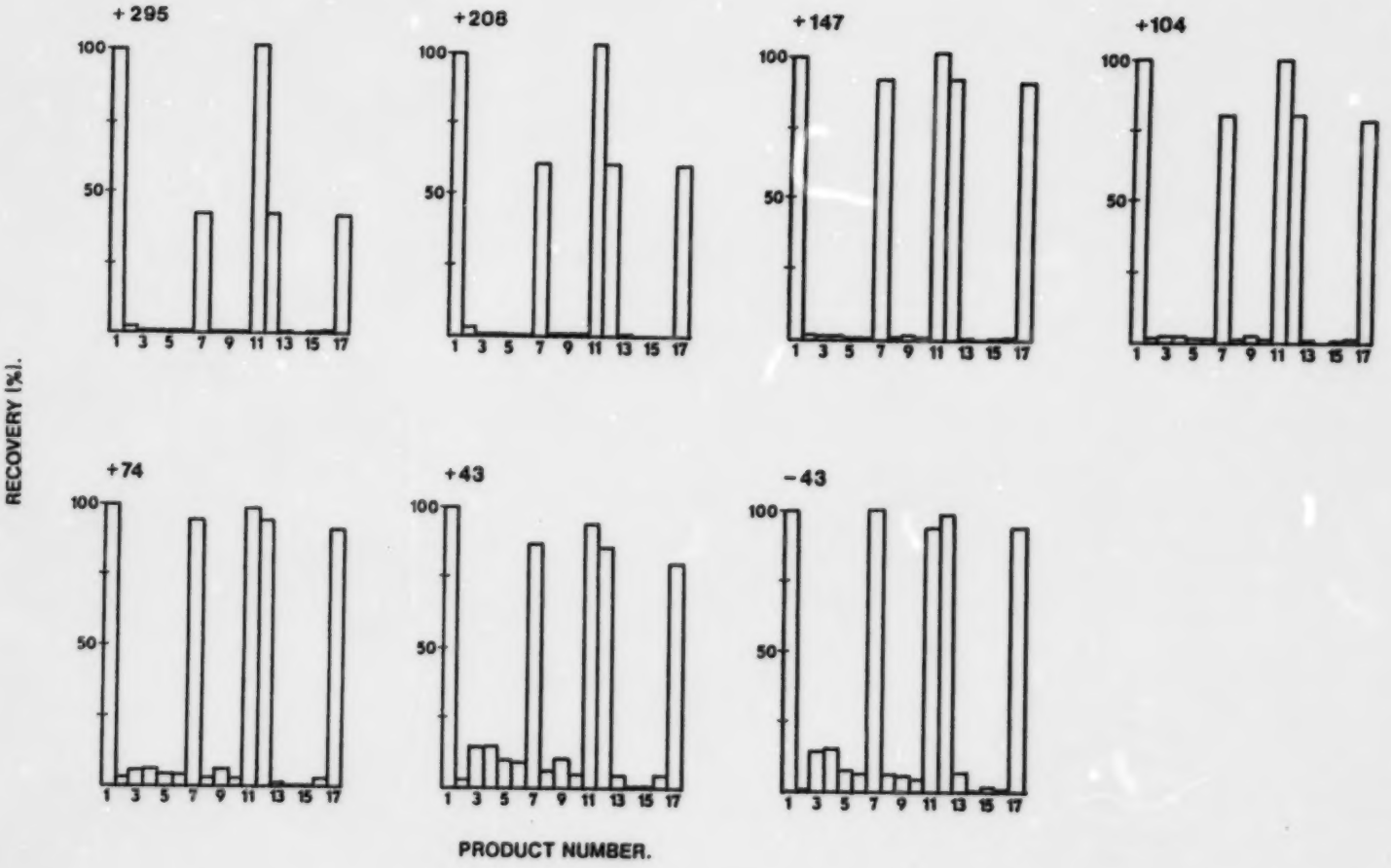


FIGURE 19.
HISTOGRAMS PORTRAYING THE RECOVERY OF THE SCREEN SIZE FRACTIONS IN THE 17 MILL PRODUCTS. PRODUCT NUMBERS AS IN FIGURES 2 AND 4, AND TABLE 45.

GRAIN SIZE ANALYSIS OF MINERALS BY IMAGE ANALYSIS

GRAIN SIZE DISTRIBUTIONS OF THE PRINCIPAL MINERALS IN THE FEED

Before dealing with the grain size distributions of free and of unliberated chalcopyrite, sphalerite and galena in the various products, it is appropriate to consider the grain size of these minerals in the feed, irrespective of their state of liberation. The grain sizes determined by image analysis were generated by measuring the surface areas of the grains and using the square root of the grain area as the diameter. This 'equivalent square diameter' (or equivalent edge length of a square) gives grain sizes that compare closely with screen analyses (Petruk 1978). As described under PETROGRAPHIC AND IMAGE ANALYSIS, the grain size distributions of the free and of the unliberated chalcopyrite, sphalerite and galena in each sample were determined, whereas that of total chalcopyrite, sphalerite and galena were not. Therefore, the grain size distributions of free and of unliberated chalcopyrite in the feed were combined to produce the grain size distribution of chalcopyrite (total). This was also done for sphalerite and galena, and the data are presented in Table 46. Also included in Table 46 is the particle size distribution of the grinding curve and the grain size distributions of sulfides (combined) and pyrite. The grain size distribution for the sulfides was generated by performing the size analysis on the combined images of all sulfides detected (by the major mineral routine) i.e. chalcopyrite, sphalerite, pyrite, galena, arsenopyrite and pyrrhotite. Unfortunately, the grain size distribution of pyrite was not experimentally determined, and so it was estimated from that of the sulfides. Because chalcopyrite, sphalerite, pyrite and galena comprise 22 %, 18 %, 58 % and 0.3 %, respectively of the sulfides in the feed, and because the grain size distributions of chalcopyrite, sphalerite and galena were determined, that of pyrite could be calculated. However, the calculated pyrite grain size distribution contains a small extraneous component due to minor pyrrhotite and trace arsenopyrite, but is considered to be essentially accurate. These grain size distributions are plotted as cumulative curves in Figure 20. The cumulative grain size curves for pyrite, chalcopyrite and sphalerite in the unbroken ore (Healy & Petruk 1988) are also included in Figure 20.

From Figure 20 it can be seen that the total sulfides lie on the slightly finer side of the grinding curve, and thus dictate that the silicate and carbonate gangue, which comprise the remainder and bulk of the feed, lie slightly on the coarse side of the grinding curve. While galena, chalcopyrite and sphalerite lie on the finer side of the grind and are thus concentrated in the finer sieve fractions, pyrite in contrast is concentrated in the coarser fractions of the grind. In fact, grinding has resulted in a reduction in the grain size of chalcopyrite and sphalerite from 80 % - 175 um and - 208 um to 80 % - 26 um and - 36 um, respectively. Furthermore, as the sulfide curve lies close to the grinding curve, that of the silicate and carbonate gangue which constitutes 80.4 vol % of the feed, must lie immediately on the coarser side of the curve i.e. approximately 80 % - 52 um. Consequently, it is probable that pyrite constitutes the coarsest of these minerals or mineral groups in the feed. The grinding has resulted in a size reduction in pyrite from

80 % - 96 um to only 80 % - 55 um. Furthermore, the general shape of the grinding curve conforms with the size distribution of pyrite in the feed, which in turn closely parallels that of pyrite in the unbroken ore. The grinding causes the size distribution of sphalerite and chalcopyrite to pivot on the finer sizes, due principally to the dramatic decrease in the maximum grain size of these minerals, suggesting strong random breakage of coarse-grained chalcopyrite and sphalerite.

In contrast, the size distribution of pyrite does not appear to pivot on the finer size ranges, with only a small decrease in the maximum grain size, suggesting that pyrite breakage is dominated by preferential breakage, with negligible random breakage. Pyrite being a 'hard' mineral is resistant to random breakage, which operates by energy intensive application of stress to the strong interatomic bonds within the mineral lattice. Thus, once having been ground by the process of preferential breakage (involving the breakage along grain boundaries, fractures and other rational crystallographic planes) to the size distribution of pyrite in the unbroken ore, the pyrite unlike chalcopyrite and sphalerite, strongly resists further breakage. The grain size distribution of pyrite in the unbroken ore was generated on images of pyrite on which boundary enhancement was performed, in order to define grain boundaries, fractures and cleavages (Healy & Petruk 1988). Thus, this grain size distribution incorporates the uninterrupted chord lengths between crystallographic features where preferential breakage may occur. The small size reduction in pyrite due to grinding implies minimal random breakage and hence negligible liberation of inclusions in pyrite. Significant amounts of galena, sphalerite, chalcopyrite and electrum (i.e. type 1) all occur as inclusions in pyrite (Healy & Petruk 1988). Of these only the fine-grained inclusions of type 1 electrum are likely to constitute significant losses of the corresponding mineral. However, much of type 1 electrum occurs as fracture fillings and grain coatings that are expected to be liberated by preferential breakage. Textures such as dendritic replacements of sphalerite and/or galena in pyrite or atoll-textured sphalerite in pyrite were also incorporated into the size distribution of the pyrite in the unbroken ore. However, as these features are not concentrated on grain boundaries or fractures and may be encapsulated by pyrite without discontinuities, their behaviour during grinding is difficult to predict. Atoll-textured sphalerite in pyrite is observed in the mill product samples. Persistent dendritic-textured sphalerite and/or galena in pyrite were not observed petrographically, though this texture is only pervasive in the No. 5 lens of the south zone, which did not contribute to the present float feed.

The grain size distribution of pyrite in the float feed is independent of its state of liberation, where individual pyrite grains may occur in composite particles with other minerals. With the exception of in the Cu-concentrate, the liberation of pyrite was not determined by image analysis. Petrographic examination indicates that approximately 60 to 70 % of the pyrite in the feed is free, though much of what was classed as free may be locked with gangue, which is difficult to distinguish from the mounting medium in reflected light. The most common types of particles in which unliberated pyrite occurs are (1) one or more (clusters) of pyrite grains with interstitial wedges and films of chalcopyrite, and (2) pyrite grains encapsulated in coarse-grained sphalerite. Clearly, as pyrite is not completely liberated, and yet size reduction has occurred, random breakage has commenced prior to completion of preferential breakage.

TABLE 46

GRAIN SIZE DISTRIBUTIONS OF CHALCOPYRITE, SPHALERITE, GALENA AND TOTAL SULFIDES IN THE FLOAT FEED.

GRAIN DIAMETER (um)	CHALCOPYRITE		SPHALERITE		GALENA		SULFIDES		PYRITE	GRINDING CURVE
	Freq.	Cum.	Freq.	Cum.	Freq.	Cum.	Freq.	Cum.	Cum.	Cum.
1.62	----	----	----	----	0.76	0.76	----	----	----	----
2.30	----	----	----	----	2.06	2.82	----	----	----	----
3.25	0.11	0.11	0.00	0.00	4.74	7.56	2.93	2.93	4.87	----
4.60	1.80	1.91	0.44	0.44	4.57	12.1	2.72	5.65	8.64	----
6.50	7.40	9.31	4.36	4.80	10.0	22.1	4.10	9.75	11.4	----
9.20	13.9	23.2	10.1	14.9	19.3	41.4	6.08	15.8	13.2	----
13.0	15.8	39.0	12.8	27.7	20.7	62.1	8.59	24.4	17.8	----
18.5	14.3	53.3	18.7	46.4	20.8	82.8	10.6	35.0	24.5	----
26.0	19.4	72.7	11.1	57.5	17.2	100.0	14.8	49.8	38.7	----
36.8	18.1	90.8	17.8	75.3			13.8	63.6	49.7	----
52.0	8.85	100.0	11.3	86.6			13.7	77.3	66.0	----
74.0			13.3	100.0			13.8	91.1	85.3	81.5
104.							5.09	96.2	93.9	89.4
147.							3.65	100.0	100.0	95.4
208.									----	98.5
+208									----	100.0

NOTE : Pyrite grain size distribution was not experimentally determined, but was calculated from that of the total sulfides (See Text).

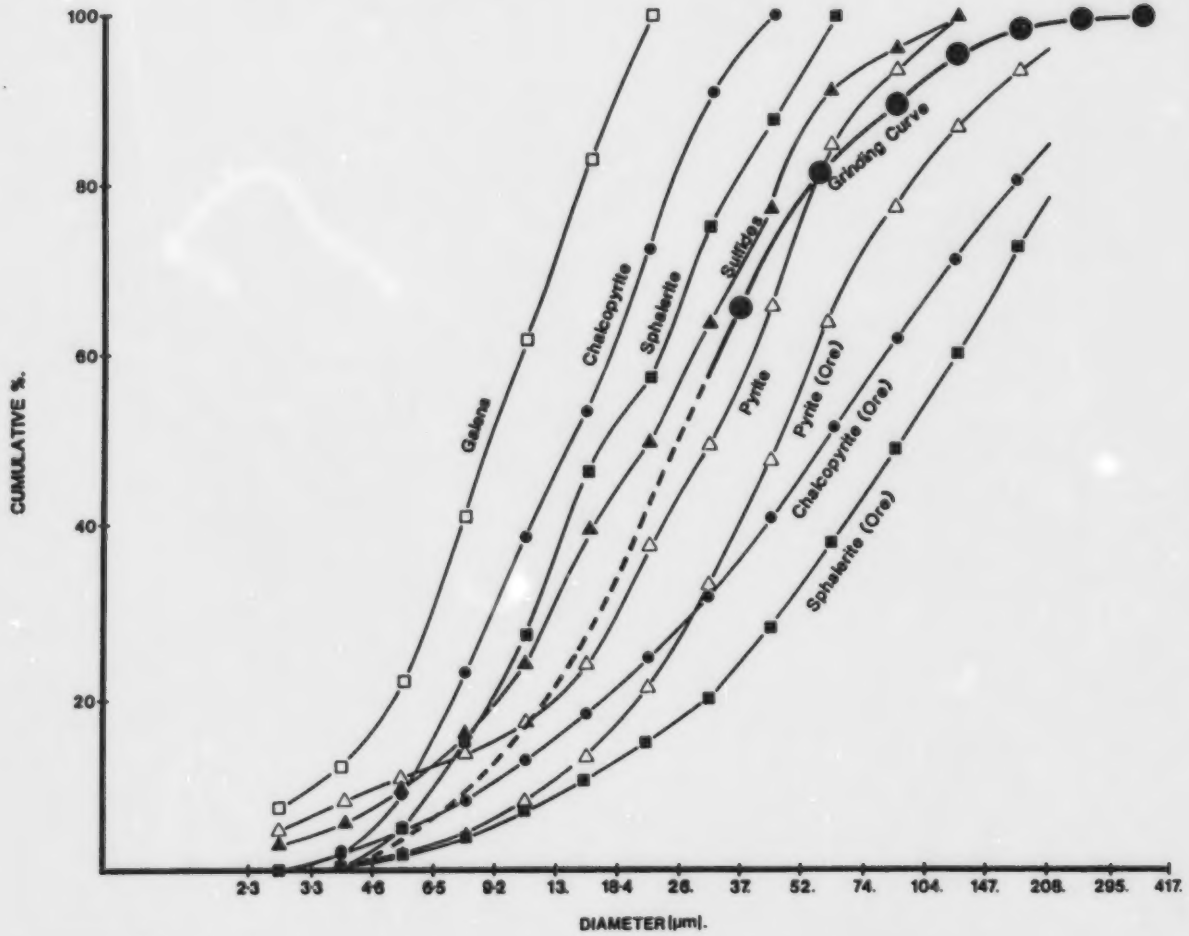


FIGURE 20.
THE GRAIN SIZE CURVES OF THE PRINCIPAL MINERALS AND MINERAL GROUPS, AND THE PARTICLE SIZE CURVE OF THE FLOAT FEED. THE GRAIN SIZE CURVES OF CHALCOPYRITE, SPHALERITE AND PYRITE IN THE UNBROKEN ORE ARE ALSO SHOWN.

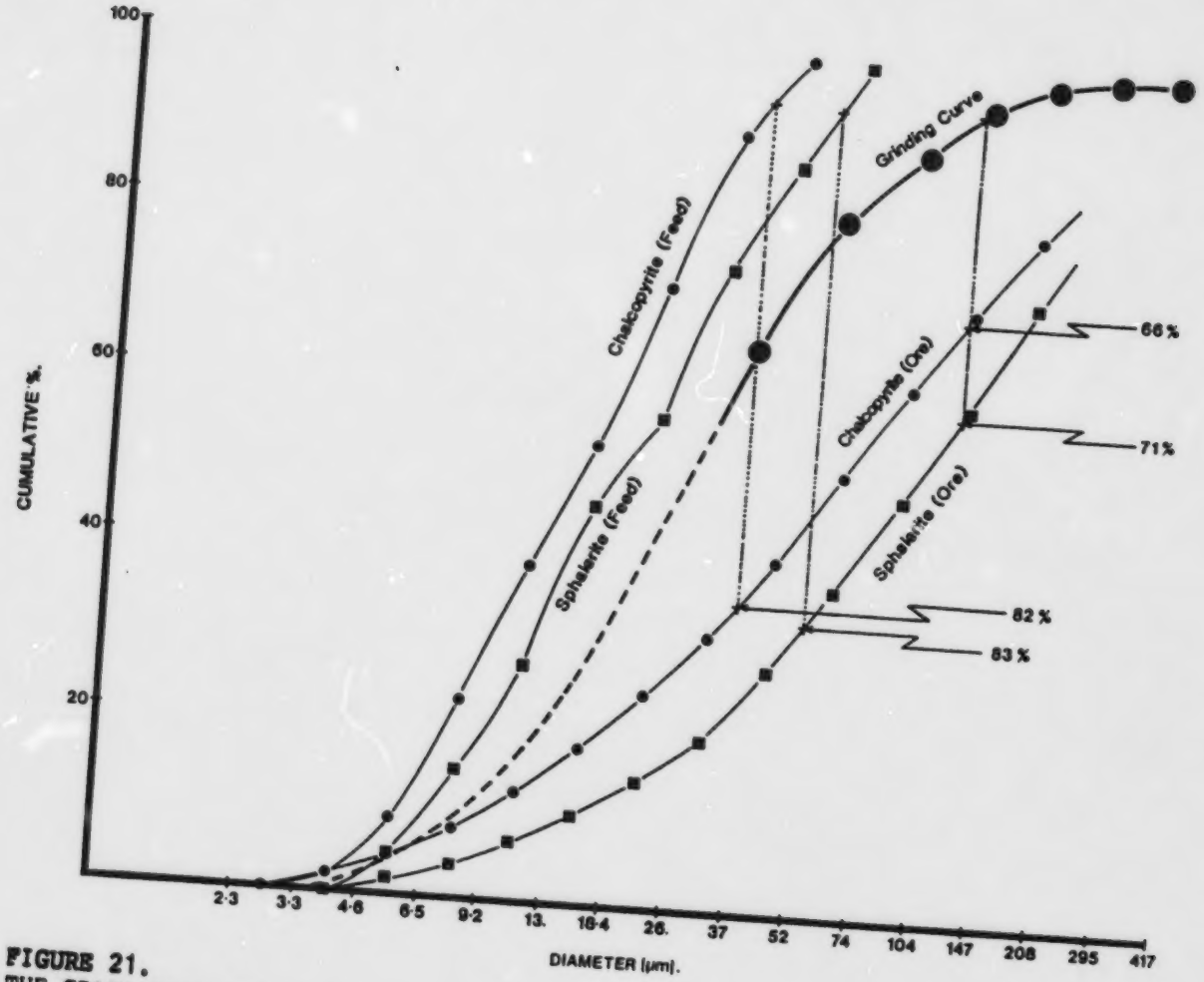


FIGURE 21.
THE GRAIN SIZE CURVES FOR CHALCOPYRITE AND SPHALERITE IN THE UNBROKEN ORE, AND IN THE FLOAT FEED. PREDICTED LIBERATIONS BASED ON THE PARTICLE SIZE DISTRIBUTION OF THE FEED (I.E. 66 % and 71 %, RESPECTIVELY), AND BASED ON THE RESPECTIVE GRAIN SIZE DISTRIBUTIONS IN THE FEED (I.E. 82 % and 83 %, RESPECTIVELY) ARE SHOWN.

In Figure 21 the grain size distributions of chalcopyrite and sphalerite in both the unbroken ore and the float feed, as well as the grinding curve are reproduced from Figure 20. Healy & Petruk (1988) predicted using Petruk's liberation model (Petruk 1976) that 62 % of the chalcopyrite and 66 % of the sphalerite would be liberated if the ore was ground to 65 % - 43 um, and where the 95 % passing size of the grind was 147 um. The raw screen size analysis of the float feed for this study indicated a particle size distribution defined by 68 % - 43 um, while the normalised, adjusted screen sizes indicate a particle size distribution defined by 66 % - 43 um, and a 95 % passing size of 115 um (See Table 41 and 44). Using this 95 % passing size of 115 um, the predicted liberation is 66 % for chalcopyrite and 71 % for sphalerite, which is considerably higher still than the material balance adjusted liberations of 60.6 % for chalcopyrite and 50.5 % for sphalerite. However, Petruk's model assumes that each mineral in the feed has equal hardness and grindability, and that random and preferential breakage are equally represented. The latter is implicit in the assumption that only 50 % liberation is achieved when the ore is ground to the size distribution of the mineral of interest. If breakage was solely by preferential breakage, then 100 % liberation would be achieved on grinding to the size distribution of the mineral of interest, and below which random breakage necessarily commences. Clearly, preferential breakage dominates initially, with progressively greater random breakage occurring as grinding to finer sizes proceeds. In addition, the minerals have different grindabilities such that they are variously concentrated in the coarser (e.g. pyrite) or finer (e.g. chalcopyrite and sphalerite) sieve fractions of the feed (Petruk et al. 1885).

Petruk's model assumes that the size distribution of all the minerals lies exactly on the grinding curve, yet the size distributions of chalcopyrite and sphalerite are finer than the grinding curve. Thus, the grinding curve is not a reasonable estimate of the size of these minerals in the ground product, and hence their state of liberation cannot be confidently predicted. If the observed size distribution of chalcopyrite in the feed is used as the grinding curve for chalcopyrite (which implicitly it is) the predicted liberation for chalcopyrite is 82 % (See Figure 21). Similarly, the predicted liberation for sphalerite is 83 %. These predicted liberations of 82 % and 83 % are thus even further amiss of the adjusted observed liberations of 60.6 % and 50.5 % for chalcopyrite and sphalerite, respectively. However, these values compare closely with those of the '90-100' or 'essentially free' classes for chalcopyrite (83 %) and sphalerite (80 %). Perhaps, it is unreasonable to define the liberation on the basis of a grain being absolutely free or not, as a 100 um diameter grain of sphalerite with a single 2 um bleb of chalcopyrite (i.e. persistant chalcopyrite-disease) is for all intents and purposes free. It is also unlikely that minerals which are characterised by abundant fine-grained inclusions (e.g. sphalerite with abundant chalcopyrite-disease), will achieve the same degree of absolute liberation as minerals not characterised by abundant inclusions (e.g. chalcopyrite), but which have a comparable grain size distribution in the unbroken ore.

GRAIN SIZE DISTIBUTIONS OF FREE, UNLIBERATED AND SUB-FREE CHALCOPYRITE, SPHALERITE AND GALENA

The raw grain size distributions for free chalcopyrite, sphalerite and galena in the 25 mill product samples are given in Tables 1, 2 and 3 of appendix C, and are condensed for the 17 principal mill products in Tables 47, 52 and 57, respectively. Similarly, the raw grain size distributions for unliberated chalcopyrite, sphalerite and galena in the 25 mill product samples are given in Tables 4, 5 and 6 of appendix C, and are condensed for the 17 principal mill products in Tables 49, 54 and 58, respectively. The total surface area of the free grains of the minerals of interest, upon which the respective grains size distribution is based, is also given in each of these twelve tables. For a given mineral, and assuming relatively comparable grain size distributions for free or unliberated grains of that mineral in each product, the total surface area is a measure of the statistical uncertainty in the measured value. Thus, in the Cu-rougher concentrate, the 1,181,000 um^2 total surface area of free chalcopyrite corresponds to approximately 3,000 grains. Similarly, the 27 um^2 total surface area of free chalcopyrite grains in the Zn-concentrate corresponds to only 2 grains.

The condensed grain size distributions of unliberated chalcopyrite, sphalerite and galena in the 17 mill products, given in Tables 49, 54 and 58, were not material balanced. As a substantial fraction of the unliberated grains of the mineral(s) of interest occur as middling and sub-middling grains, their flotation behaviour is partly controlled by the dominant accompanying mineral in the host particle, and not by the grain size of the mineral of interest. As sub-free grains are nominally expected to behave independently of the character of the accompanying subordinate locked mineral grain, deviations in flotation behaviour may be related to grain size of the mineral(s) of interest. Consequently, the grain size distribution of sub-free chalcopyrite and sphalerite, but not galena, were material balanced using MATBAL in order to simulate the behaviour of sub-free grains through the circuit on the basis of grain size.

As with other types of image analysis data that were material balanced, the Cu, Zn and Pb assays are taken as the more precise equivalents of the modal amounts of chalcopyrite, sphalerite and galena, respectively. Consequently, for each product the Cu-assay was multiplied by the modal fraction of free chalcopyrite occurring in each size class from - 3.25 um to - 104 um. The same was done for Zn and sphalerite and Pb and galena, and similarly for sub-free chalcopyrite and sphalerite. These values are transformed by MATBAL into a set of adjusted modal fractions in each grain size class of the respective free mineral and sub-free mineral, and recoveries of the same are calculated. The latter are given in Tables 55 to 59. Because MATBAL balances the individual size classes independently through the circuit, rather than balancing the grain size distributions as a whole through the circuit, the behaviour of individual classes are compared. Thus, any systematic trends in the recovery of finer size classes (e.g. slimes) or of coarser size classes (e.g. over-sized) of free and/or sub-free grains of the mineral(s) of interest may be identified. Unfortunately, the number of grains observed in the size classes of the fine-tail (i.e. - 4.60 and particularly - 3.26 um) and of the coarse-tail (i.e. - 208, - 147, - 104 and - 74 um) are low in many of the products, with resulting poor counting statistics. The initial balances achieved for the

- 3.25 um, - 4.60 um, - 74 um and - 104 um classes were characterised by erratic, non-sytematic balances. This tends to obscure any trends that might be apparent in the coarse- and fine-tails of the size distributions, and is particularly unfortunate as it is these classes of grains that lie outside the optimum size range for flotation (i.e. 5 to 100 um). These classes are most likely to exhibit anomalous flotation behaviour, which is the object of this investigation. Consequently, the - 3.26 and the - 4.60 um classes were combined, as were the - 208, - 147, - 104 and - 74 um classes (i.e. + 52.0 um class), and are nominally treated as being representative of the slime and over-sized classes, respectively. However, the balances for all classes are subject to considerable uncertainty due to poor counting statistics in all, or certain products in which these are depleted (e.g. maximum observed grain size of free chalcopyrite in the feed is 52 um, but is 208 um in the Cu-concentrate).

All the material balances presented in this report, with the exception of those on the sieve size analyses of the mill products (generated by J. Wilson and W. Petruk at CANMET) did not incorporate a 'non-normal' node to account for the regrind between the Cu-rougher and the Cu-scavenger. While in the latter balances the failure to account for the regrind is not considered significant, it is significant in modelling the behaviour of grain sizes of free and sub-free grains of the mineral(s) of interest. However, the author was unable to implement the 'non-normal' node function, such that the change in grain sizes at the regrind are not incorporated into the material balance model. Consequently, an incomplete simulation of the grain sizes through the circuit was achieved, and probably contributed to the unsatisfactory balances generated for both the coarse- and fine-tails of the size distributions. In general, the material balance modelling of the grain size classes are unsatisfactory, and do not lend themselves to confident interpretation.

CHALCOPYRITE

The raw grain size distributions of free, unliberated and sub-free chalcopyrite in the 17 mill products are given in Tables 47, 49 and 50, respectively. The material balance calculated recoveries of the grain size classes of free and sub-free chalcopyrite are given in Tables 48 and 51. Plots of the raw grain size distribution of free chalcopyrite and of the grain size distribution and classification of unliberated chalcopyrite in the feed, Cu-concentrate and Zn-concentrate are given in Figures 1 to 3 of appendix C.

FREE GRAINS

In the chapter on MINERAL LIBERATIONS it was detemined by material balance calculation that 99.3 % of the free chalcopyrite is recovered in the Cu-concentrate. This excellent recovery implies that all grain size classes of free chalcopyrite are completely recovered with negligible losses. A discussion of free chalcopyrite, as measured by image analysis is thus essentially limited to the Cu-circuit products. However, it is also noted that some coarse-grained (or over-sized) free and sub-free chalcopyrite report to the Zn-concentrate as observed petrographically (See Plate 2A and 2B). This was not observed by image analysis, because of incomplete coverage of the

inhomogeneous polished sections of the Zn-circuit concentrates. Consequently, it was not incorporated into the material balance modelling of the behaviour of, and particularly of the coarser classes of, free chalcopyrite through the circuit.

The grain size curves for free chalcopyrite in the feed and Cu-circuit concentrates are presented in Figure 22. Three significant observations are apparent. Firstly, free chalcopyrite in all the Cu-circuit concentrates are coarser-grained than in the feed, suggesting that either fine-grained chalcopyrite is lost to the tailings, or most probably that the data for the feed is somewhat inaccurate. Secondly, there is a trend of progressively finer grain sizes of free chalcopyrite in the Cu-rougher and cleaner concentrates, suggesting that coarse-grained free chalcopyrite is partly rejected during Cu-cleaning. Thirdly, the grain size of free chalcopyrite in the Cu-scavenger concentrate is the coarsest in these products, though the maximum measured grain size is less than 104 um. Thus, the Cu-1st cleaner rejects abundant coarse-grained chalcopyrite, whereas the 'hard-pulling' Cu-scavenger recovers this fraction. This confirms that coarse-grained free chalcopyrite is not readily floatable, is concentrated in the two products that comprise the proposed regrind stream, and is partly lost to the Zn-circuit.

Material balance calculated recoveries of free chalcopyrite grain size classes gave values between 95.6 and 99.6 % in the Cu-concentrate (See Table 48). The lower recoveries (i.e. < 98 %) correspond to the - 4.60 and the + 52.0 um classes, and reflect either a real fall-off in flotation efficiency, and/or poor balances for these classes of free chalcopyrite. It is apparent from Table 48 that the recoveries and/or circulating loads of free chalcopyrite at the Cu-rougher and Cu-1st cleaner increase with grain size such that the Cu-cleaner feed contains 163.1 % of the free chalcopyrite in the + 52.0 um class, while 53.2 % is rejected by the Cu-1st cleaner (See Figure 23 and Plate 2E). Thus, while free chalcopyrite is largely recovered, the efficiency in flotation drops off with increasing grain size, with losses of over-sized free chalcopyrite to the Zn-circuit, principally to the Zn-concentrate. Regrinding the Cu-1st cleaner tailings and the Cu-scavenger concentrate, where the latter was not indicated by material balancing of the image analysis data, is expected to address these losses by reducing the maximum grain size of free chalcopyrite into the optimum flotation range of < 100 um.

UNLIBERATED GRAINS

In the chapter on MINERAL LIBERATIONS it was determined by material balance that 79.7 % of the unliberated chalcopyrite is recovered in the Cu-concentrate, with significant losses to the Zn-concentrate (i.e. 8.52 %) and the tailings (i.e. 11.8 %). These losses are potentially controlled by the degree of liberation of the chalcopyrite, and both the grain size of the particle and of the constituent grains of unliberated chalcopyrite. Thus, material balance modelling the behaviour of these grains through the circuit on the basis of grain size was not performed, as any observed trend would incorporate the affects of these other parameters. However, as 75.6 % of the unliberated chalcopyrite consists of sub-free chalcopyrite, both the liberation of the grain and the grain size of the host particle, and hence it's flotation behaviour, approximate to that of an equivalent free grain of chalcopyrite.

The grain size distributions of unliberated chalcopyrite are given in Table 49, and those of the principal products of interest are plotted in Figure 24. It is apparent that there is no significant fractionation of unliberated chalcopyrite on the basis of grain size between the feed, the Cu-concentrate and the tailings. By contrast, the unliberated chalcopyrite in the Zn-concentrate is preferentially finer-grained, and principally reflects chalcopyrite-disease, fine-grained diablastic and granular sphalerite-chalcopyrite particles, and middling pyrite-chalcopyrite and pyrrhotite-chalcopyrite particles (See Plate 2A and 2D). The maximum grain size of unliberated chalcopyrite in the Zn-concentrate is less than 37 um, which does not include the rare, but volumetrically significant over-sized sub-free grains of chalcopyrite petrographically observed, but which were not observed by image analysis. Thus, in contrast to that measured, the actual grain size of unliberated chalcopyrite in the Zn-concentrate is bimodal, though the volume proportion occurring as over-sized sub-free chalcopyrite is undetermined. The grain size distribution of unliberated chalcopyrite in the Cu-scavenger concentrate is 80 % - 76 um, and as with free chalcopyrite, is strikingly coarser than that of any other product. This again emphasizes the difficulty of floating over-sized free and unliberated chalcopyrite, and it's preferential concentration in the Cu-scavenger concentrate.

The grain size distributions of sub-free chalcopyrite are given in Table 50, while the material balance calculated recoveries of the grain size classes are given in Table 51. No significant losses of any grain size class of sub-free chalcopyrite to the tailings are indicated in Table 50, and contrast with the material balance calculated recoveries of the finer grain size classes in the tailings i.e. 29.8 % of the -4.60 um class (See Table 51). However, this is in general agreement with the 91.2 % recovery of sub-free chalcopyrite in the Cu-concentrate and the 7.21 % recovery in the tailings (See Table 28). Thus, the minor losses of sub-free chalcopyrite to the tailings may consist preferentially of finer-grained size classes. While petrographic and image analysis of the tailings failed to identify losses of fine-grained sub-free chalcopyrite, the loss of part of such a slime fraction is indicated by material balance modelling of the grain size classes of sub-free chalcopyrite through the circuit (See Figure 25). Also apparent from Table 51 is the increasing circulating loads in the Cu-circuit of sub-free chalcopyrite with increasing grain size, and the conspicuous recoveries of 90 and 88 % of the + 52 um class in the Cu-scavenger and Cu-1st cleaner tailings, respectively (See Figure 25).

SPHALERITE

The raw grain size distributions of free, unliberated and sub-free sphalerite in the 17 mill products are given in Tables 52, 54 and 55, respectively. The material balance calculated recovery of the grain size classes of free and sub-free sphalerite are given in Tables 53 and 56, respectively. Plots of the raw grain size distribution of free sphalerite and of the grain size distribution and classification of unliberated sphalerite in the feed, Cu-concentrate and Zn-concentrate are given in Figures 4 to 6 of appendix C.

TABLE 47

GRAIN SIZE DISTRIBUTIONS OF FREE CHALCOPYRITE IN THE 17 MILL PRODUCTS.

	Product.	Classes (um). -3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	Area. (um^2)
1	Float Feed.	0.07	2.12	8.81	18.2	20.9	12.5	19.8	11.4	6.28	0.00	0.00	0.00	0.00	34600.
2	Cu Scav Co.	0.49	1.12	1.56	2.39	5.15	4.36	7.79	6.98	26.7	18.9	24.5	0.00	0.00	66700.
3	Cu Roug Co.	0.26	1.24	3.81	6.30	10.2	13.6	16.3	17.9	15.0	13.1	2.03	0.00	0.00	1181000.
4	Cu Cl 1 Fd.	0.09	0.87	3.53	6.33	9.21	13.5	15.0	21.1	20.1	7.06	3.20	0.00	0.00	604000.
5	Cu Cl 1 Co.	0.75	2.81	5.92	9.60	13.6	15.2	18.5	14.4	11.7	7.48	0.00	0.00	0.00	486000.
6	Cu Cl 2 Co.	1.55	4.35	7.68	12.0	16.9	18.6	15.4	10.7	4.77	0.40	0.00	0.01	7.46	636000.
7	Zn Roug Fd.	15.9	84.1	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	32.
8	Zn Roug Co.	0.69	3.48	8.54	10.9	10.8	13.9	8.80	15.1	5.19	7.12	15.4	0.00	0.00	30279.
9	Zn Cl 1 Co.	0.11	1.47	4.40	12.7	12.3	22.8	0.00	25.8	20.4	0.00	0.00	0.00	0.00	7600.
10	Zn Cl 2 Co.	0.00	31.7	68.3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	27.
11	Cu Roug Tl.	0.00	2.23	8.83	5.70	7.28	16.0	59.9	0.00	0.00	0.00	0.00	0.00	0.00	2190.
12	Cu Scav Tl.	0.70	5.28	9.59	23.5	17.4	16.3	12.0	15.2	0.00	0.00	0.00	0.00	0.00	5405.
13	Cu Cl 1 Tl.	0.15	0.94	3.96	6.40	10.8	12.1	14.1	15.2	17.1	13.7	5.63	0.00	0.00	173800.
14	Cu Cl 2 Tl.	0.41	4.26	10.3	15.8	17.9	16.1	14.4	9.42	3.64	4.33	3.36	0.00	0.00	136000.
15	Zn Cl 1 Tl.	0.00	6.13	18.7	28.0	17.9	0.00	29.3	0.00	0.00	0.00	0.00	0.00	0.00	962.
16	Zn Cl 2 Tl.	0.00	2.30	6.54	11.2	15.6	17.2	0.00	23.0	24.3	0.00	0.00	0.00	0.00	5170.
17	Zn Roug Tl.	0.00	10.3	19.8	0.00	69.8	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	101.

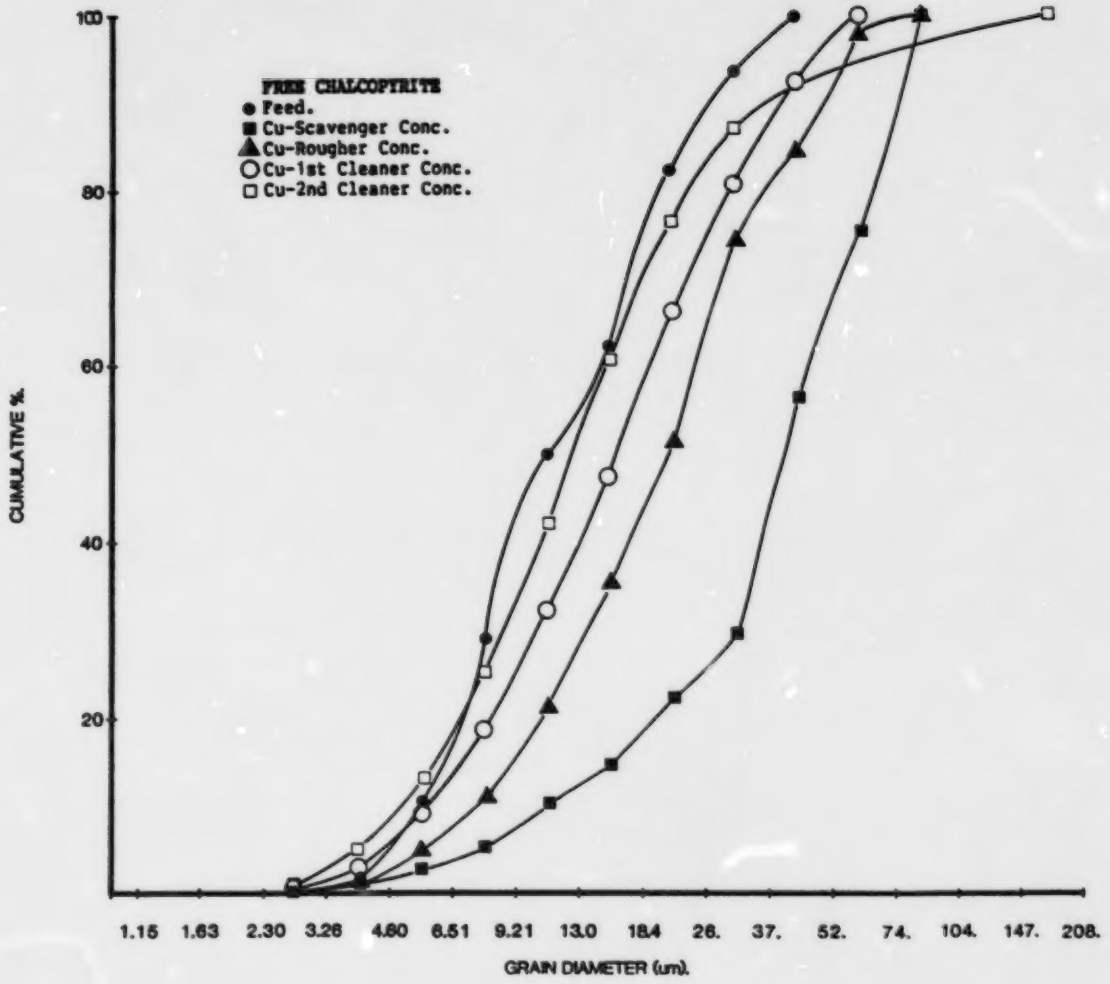


FIGURE 22.
THE GRAIN SIZE CURVES FOR FREE CHALCOPYRITE IN SELECTED MILL PRODUCTS.

TABLE 48

RECOVERY OF GRAIN SIZE CLASSES OF FREE CHALCOPYRITE.

	Product.	TOTAL	Classes (um). -4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0
1	Float Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2	Cu Scav Co.	2.65	2.44	0.96	0.88	0.93	0.92	1.34	0.21	0.16	2.11
3	Cu Roug Co.	107.5	101.8	105.0	105.5	106.6	106.4	106.5	109.4	121.9	150.6
4	Cu Cl 1 Fd.	112.9	117.9	116.5	115.9	114.5	112.5	111.1	113.2	124.8	163.1
5	Cu Cl 1 Co.	104.6	111.7	109.8	109.5	107.3	105.5	104.2	103.3	101.9	109.8
6	Cu Cl 2 Co.	99.3	95.6	98.3	99.1	99.5	99.4	99.6	99.5	99.0	97.4
7	Zn Roug Fd.	0.88	4.74	1.99	1.19	0.70	0.58	0.49	0.53	1.05	2.68
8	Zn Roug Co.	0.18	0.42	0.38	0.29	0.19	0.04	0.10	0.04	0.08	0.19
9	Zn Cl 1 Co.	0.37	0.44	0.42	0.55	0.43	0.48	0.03	0.64	1.17	0.16
10	Zn Cl 2 Co.	0.04	0.07	0.10	0.04	0.03	0.03	0.02	0.29	0.06	0.11
11	Cu Roug Tl.	3.38	6.82	2.67	1.83	1.46	1.49	1.74	0.74	1.20	4.75
12	Cu Scav Tl.	0.73	4.38	1.71	0.95	0.53	0.57	0.41	0.52	1.03	2.64
13	Cu Cl 1 Tl.	8.27	6.19	6.72	6.44	7.13	6.99	6.91	9.87	22.9	53.2
14	Cu Cl 2 Tl.	5.33	16.1	11.5	10.4	7.85	6.06	4.59	3.86	2.94	12.5
15	Zn Cl 1 Tl.	0.15	0.36	0.28	0.24	0.16	0.01	0.08	0.01	0.02	0.04
16	Zn Cl 2 Tl.	0.33	0.37	0.50	0.40	0.45	0.01	0.61	0.61	1.11	0.05
17	Zn Roug Tl.	0.70	4.31	0.90	0.51	0.55	0.39	0.50	0.50	0.97	2.53

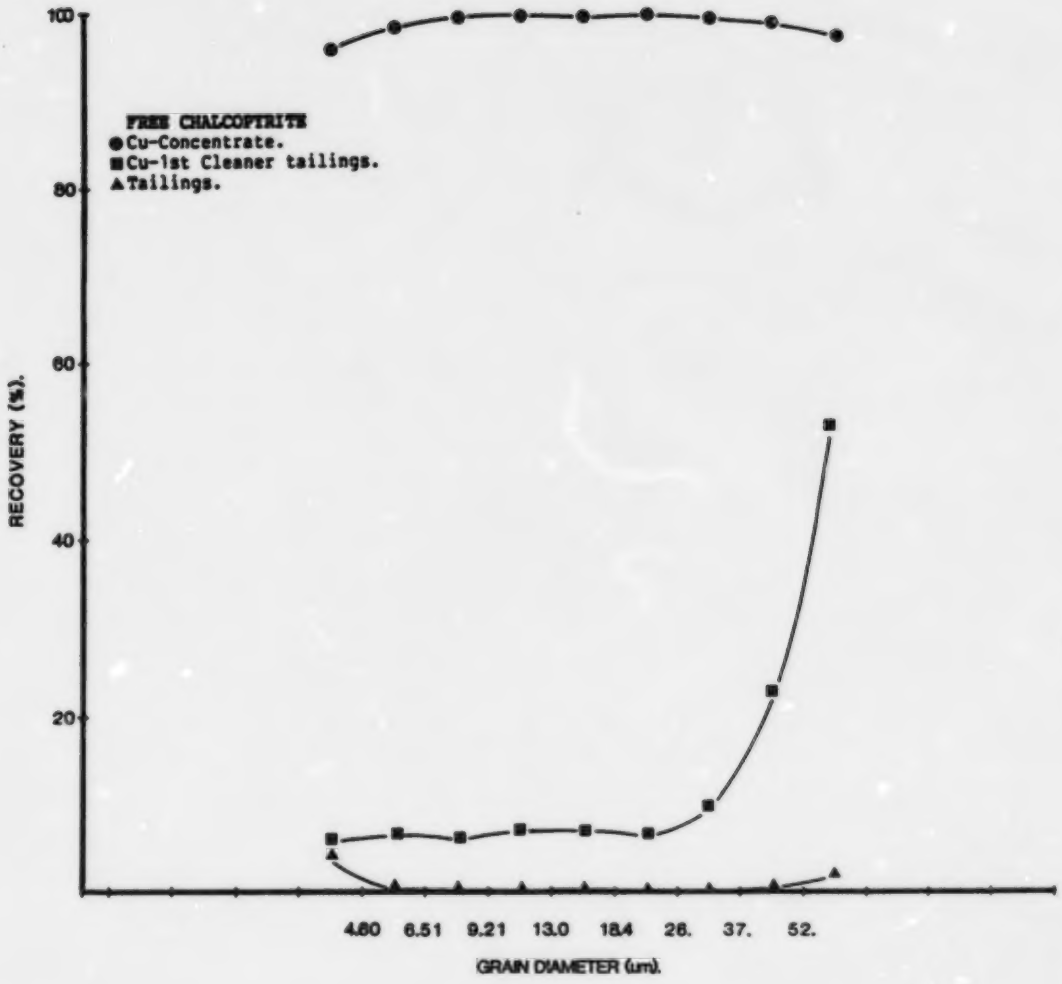


FIGURE 23.
THE RECOVERY CURVES FOR GRAIN SIZE CLASSES OF FREE CHALCOPYRITE IN SELECTED MILL PRODUCTS.

TABLE 49

GRAIN SIZE DISTRIBUTIONS OF UNLIBERATED CHALCOPYRITE IN THE 17 MILL PRODUCTS.

	Product.	Classes (um). -3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	Area. (um^2)
1	Float Feed.	0.17	1.35	7.79	9.89	16.8	18.9	27.3	12.4	0.00	0.00	0.00	0.00	0.00	25000.
2	Cu Scav Co.	0.26	0.69	1.15	2.43	4.30	5.30	9.18	8.01	18.5	19.9	15.9	14.4	0.00	295000.
3	Cu Roug Co.	0.13	0.50	1.16	2.30	4.02	6.19	10.1	15.1	21.4	28.1	9.66	1.48	0.00	780000.
4	Cu Cl 1 Fd.	0.12	0.67	1.61	2.72	3.68	7.74	10.4	16.9	27.8	14.4	13.9	0.00	0.00	329000.
5	Cu Cl 1 Co.	0.33	0.94	1.83	3.62	5.46	8.63	13.6	19.3	18.2	13.9	0.00	14.2	0.00	253000.
6	Cu Cl 2 Co.	0.81	1.87	3.38	6.20	10.5	16.9	22.0	23.1	11.0	1.17	3.03	0.00	0.00	243000.
7	Zn Roug Fd.	47.9	0.00	8.07	13.4	13.4	17.3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1680.
8	Zn Roug Co.	0.71	3.39	6.88	11.1	12.9	13.7	15.1	16.6	9.09	10.4	0.00	0.00	0.00	157540.
9	Zn Cl 1 Co.	0.04	1.60	5.37	9.10	10.9	13.7	17.2	16.7	15.6	9.75	0.00	0.00	0.00	47400.
10	Zn Cl 2 Co.	0.89	3.43	10.0	10.8	16.7	28.4	3.92	25.9	0.00	0.00	0.00	0.00	0.00	11200.
11	Cu Roug Tl.	7.63	0.63	6.36	7.04	2.40	12.6	21.9	0.00	41.4	0.00	0.00	0.00	0.00	6380.
12	Cu Scav Tl.	5.50	1.52	2.83	4.09	10.6	7.56	6.88	16.4	8.50	36.2	0.00	0.00	0.00	21010.
13	Cu Cl 1 Tl.	0.17	1.12	3.15	6.48	10.8	12.4	14.9	15.0	16.7	19.4	0.00	0.00	0.00	228400.
14	Cu Cl 2 Tl.	0.40	2.14	4.81	8.59	14.0	13.9	14.1	18.2	19.4	4.44	0.00	0.00	0.00	62700.
15	Zn Cl 1 Tl.	2.69	1.18	7.18	11.4	13.3	25.2	14.4	0.00	24.7	0.00	0.00	0.00	0.00	7620.
16	Zn Cl 2 Tl.	0.00	0.61	3.21	4.36	11.2	13.8	9.91	11.2	11.8	4.87	29.0	0.00	0.00	66100.
17	Zn Roug Tl.	1.50	1.89	0.98	9.96	9.19	6.98	26.5	43.0	0.00	0.00	0.00	0.00	0.00	2480.

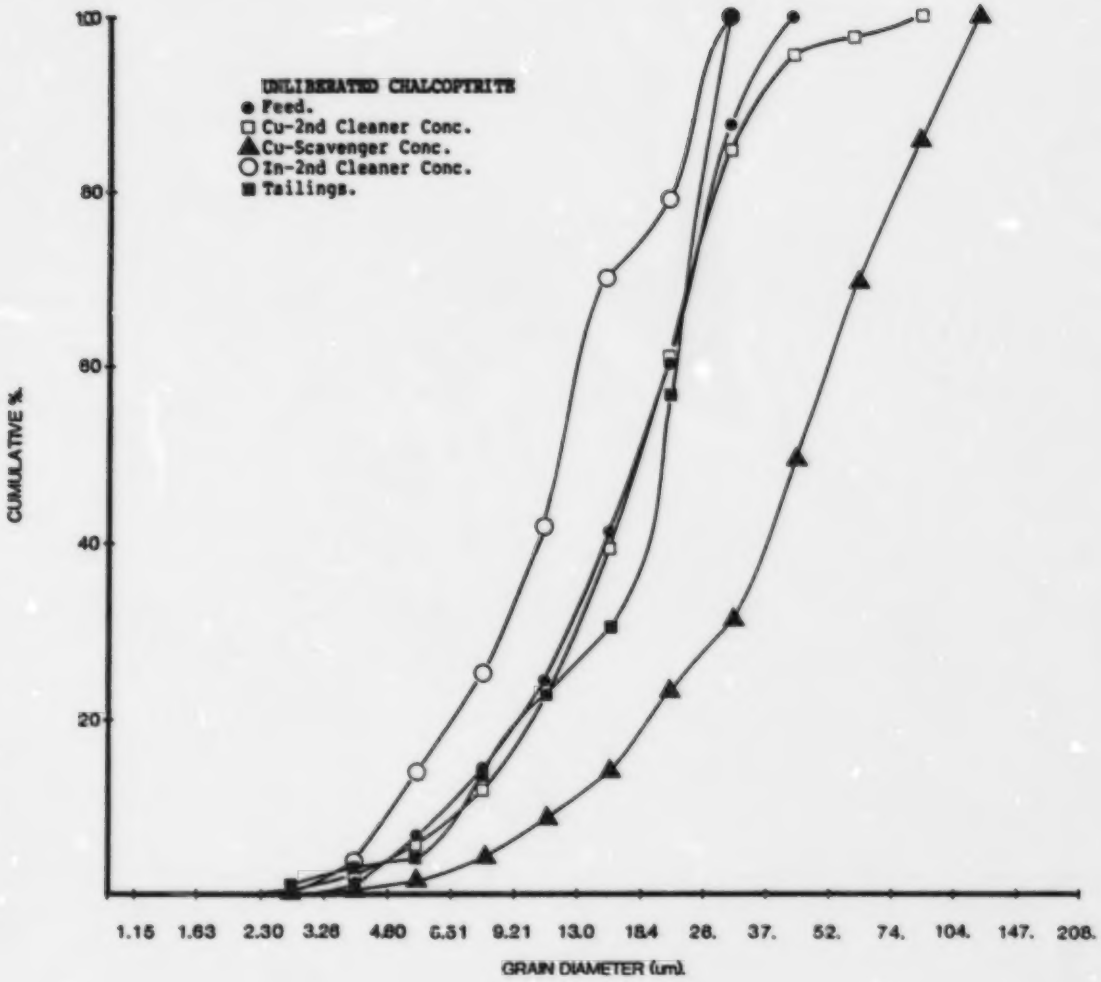


FIGURE 24.
THE GRAIN SIZE CURVES FOR UNLIBERATED CHALCOPYRITE IN SELECTED MILL PRODUCTS.

TABLE 50

GRAIN SIZE DISTRIBUTIONS OF SUB-FREE CHALCOPYRITE IN THE 17 MILL PRODUCTS.

	Product.	Classes (um). -3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	Area. (um^2)
1	Float Feed.	0.00	0.45	2.90	5.00	6.29	10.5	12.4	18.5	12.4	0.00	0.00	0.00	0.00	17040.
2	Cu Scav Co.	0.10	0.01	0.32	0.73	1.45	1.29	3.72	5.69	14.7	19.9	13.7	14.4	0.00	225720.
3	Cu Roug Co.	0.06	0.22	0.54	1.16	2.30	4.51	7.81	13.2	19.6	27.1	9.85	1.57	0.00	678220.
4	Cu Cl 1 Fd.	0.02	0.20	0.54	1.00	1.85	4.83	12.6	13.1	26.3	14.4	13.9	0.00	0.00	275470.
5	Cu Cl 1 Co.	0.13	0.44	0.99	2.39	3.42	6.39	11.7	18.5	17.5	13.9	0.00	14.2	0.00	226700.
6	Cu Cl 2 Co.	0.46	1.27	2.43	5.43	9.39	15.6	20.9	22.7	11.3	1.17	3.03	0.00	0.00	227060.
7	Zn Roug Fd.	0.24	0.00	1.46	10.7	7.16	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	330.
8	Zn Roug Co.	0.04	0.32	0.45	1.00	2.08	1.92	5.31	4.58	5.83	11.1	0.00	0.00	0.00	49345.
9	Zn Cl 1 Co.	0.00	0.06	0.86	1.40	1.91	4.51	4.20	6.26	9.37	0.00	0.00	0.00	0.00	13531.
10	Zn Cl 2 Co.	0.00	0.00	0.58	1.22	1.91	9.39	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1465.
11	Cu Roug Tl.	0.00	0.00	0.00	0.00	0.00	4.76	0.00	0.00	0.00	0.00	0.00	0.00	0.00	304.
12	Cu Scav Tl.	0.03	0.38	0.89	1.61	1.36	0.79	2.48	5.88	7.22	24.1	0.00	0.00	0.00	9256.
13	Cu Cl 1 Tl.	0.01	0.16	0.56	1.43	3.41	3.02	6.42	7.23	11.6	19.7	0.00	0.00	0.00	119850.
14	Cu Cl 2 Tl.	0.13	0.66	1.73	2.95	6.21	8.09	12.2	16.1	16.9	4.44	3.36	0.00	0.00	43580.
15	Zn Cl 1 Tl.	0.00	0.47	1.65	5.51	1.44	17.7	7.36	0.00	0.00	0.00	0.00	0.00	0.00	2610.
16	Zn Cl 2 Tl.	0.00	0.14	0.61	1.21	4.72	2.74	4.05	6.84	8.56	4.87	29.0	0.00	0.00	41490.
17	Zn Roug Tl.	0.00	0.00	0.00	0.00	0.00	0.00	0.00	43.0	0.00	0.00	0.00	0.00	0.00	1070.

NOTE : Data presented as areal percent of unliberated chalcopyrite occurring as sub-free (70-99.9%) chalcopyrite, occurring in each grain size class.

TABLE 51

RECOVERY OF GRAIN SIZE CLASSES OF SUB-FREE CHALCOPYRITE.

	Product.	TOTAL	Classes (um). -4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0
1	Float Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2	Cu Scav Co.	16.4	4.61	4.95	4.26	5.61	4.38	0.71	0.15	0.23	90.0
3	Cu Roug Co.	103.6	77.9	93.8	107.0	112.6	108.5	111.0	108.3	137.9	185.5
4	Cu Cl 1 Fd.	107.2	86.4	102.8	116.7	123.6	114.9	116.2	113.0	161.8	189.3
5	Cu Cl 1 Co.	94.8	77.3	91.0	98.1	100.2	104.6	104.5	103.8	119.9	101.0
6	Cu Cl 2 Co.	91.2	68.8	82.0	88.4	89.1	98.2	99.2	99.2	96.0	97.2
7	Zn Roug Fd.	9.63	34.0	21.2	17.9	12.2	3.73	1.83	0.86	4.04	2.90
8	Zn Roug Co.	2.42	4.25	8.13	11.3	7.66	2.03	1.09	0.06	0.27	0.34
9	Zn Cl 1 Co.	5.82	3.06	9.92	11.6	15.9	3.92	3.12	3.56	20.3	0.94
10	Zn Cl 2 Co.	1.61	1.38	4.94	5.03	6.31	0.11	0.06	0.04	0.19	0.25
11	Cu Roug Tl.	25.2	35.8	23.0	15.9	16.5	6.19	1.51	0.99	4.19	92.8
12	Cu Scav Tl.	8.82	31.2	18.0	11.7	10.9	1.81	0.80	0.84	3.96	2.81
13	Cu Cl 1 Tl.	12.5	9.12	11.9	18.6	23.4	10.3	11.8	9.15	41.8	88.3
14	Cu Cl 2 Tl.	3.60	8.49	8.99	9.69	11.1	6.43	5.26	4.67	23.9	3.85
15	Zn Cl 1 Tl.	0.81	2.87	3.19	6.28	1.34	1.92	1.03	0.02	0.08	0.10
16	Zn Cl 2 Tl.	4.21	1.68	4.98	6.56	9.62	3.81	3.06	3.52	20.2	0.70
17	Zn Roug Tl.	7.21	29.8	13.1	6.62	4.57	1.70	0.74	0.80	3.77	2.56

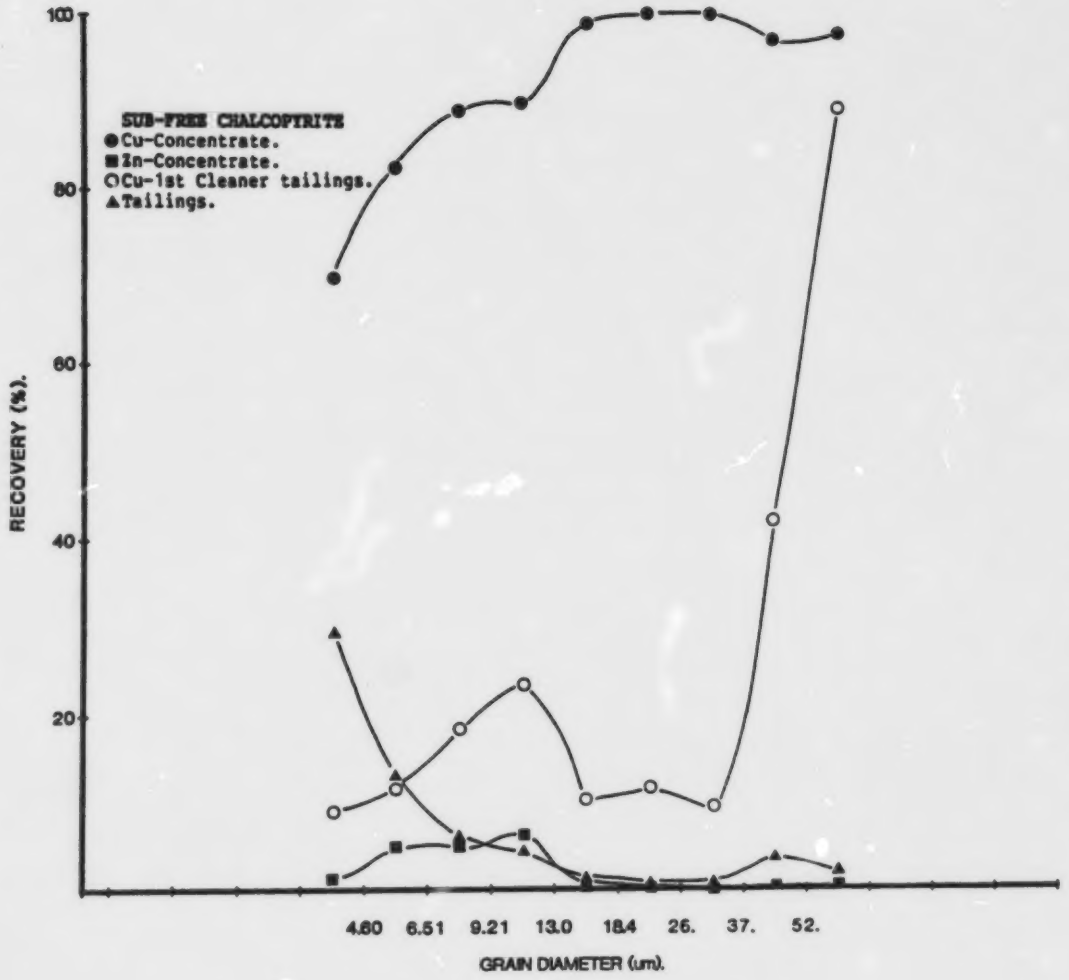


FIGURE 25.
THE RECOVERY CURVES FOR THE GRAIN SIZE CLASSES OF SUB-FREE CHALCOPYRITE IN SELECTED MILL PRODUCTS.

FREE GRAINS

In the chapter on MINERAL LIBERATIONS it was determined by material balance calculation that 90.7, 9.30 and 0.01 % of the free sphalerite is recovered in the Zn-concentrate, Cu-concentrate and tailings, respectively (See Table 21). As the nature and degree of the liberation is not a controlling factor in the behaviour of free grains of sphalerite, grain size may exercise significant control on those grains of free sphalerite that are lost to the Cu-concentrate.

From the raw data in Table 52 and the corresponding curves in Figure 26, it is apparent that free sphalerite in the feed (80 % - 22 um) is fractionated on the basis of grain size into the Zn-concentrate (80 % - 31 um) and the Cu-concentrate (80 % - 17 um). However, the slime (i.e. - 5 um) fraction constitutes only 13.5 % of the free sphalerite in the Cu-concentrate. The erratic nature of the recovery curves of free sphalerite in the Cu- and Zn-concentrate shown in Figure 28, most probably reflect the poor counting statistics in each class of free sphalerite. However, the general trend of increasing recovery of free sphalerite with decreasing grain size to the Cu-concentrate is interpreted to reflect real variation in the flotation behaviour of free sphalerite grains. Furthermore, in the Cu-cleaning circuit, the minor circulating loads of free sphalerite increase with decreasing grain size (See Table 53). Thus, while the entrainment of sphalerite slimes may be a minor contributing factor in the Zn losses to the Cu-concentrate, the finer grain size suggests a related hydraulic control affecting the flotation behaviour of free sphalerite grains. This effect is not observed for free grains of the inert mineral pyrite (See THE BEHAVIOUR OF PYRITE IN THE Cu-CIRCUIT), suggesting that sphalerite is not entirely inert prior to activation with $CuSO_4$. It is also noteworthy that the Cu-scavenger preferentially recovers relatively coarse-grained free sphalerite (See Figure 26).

The grain size curves of free sphalerite in the Zn-circuit feed and concentrates are plotted in Figure 27. The curve for the feed is finer than that of any of these products, suggesting that the finer classes are partly lost to the tailings and/or that the circulating load during Zn-cleaning consists preferentially of coarse-grained sphalerite. Fine-grained free sphalerite was not observed during either petrographic or image analysis of the tailings, but minor losses of the finer classes (e.g. 6.33 % of the - 4.60 um class) are calculated by material balances (See Table 53 and Figure 28). The finer grain size of free sphalerite in the Zn-concentrate compared to the Zn-rougher and Zn-1st cleaner concentrates (See Figure 27) suggests minor circulating loads of coarse-grained sphalerite. However, material balance calculations indicate the maximum recovery of free sphalerite to the Zn-concentrate is achieved for the coarser size classes i.e. 99.3 and 99.8 % in the - 52 and + 52 um classes (See Table 53). It is concluded that coarse-grained free sphalerite that reports to the Zn-cicruit is completely recovered, but that minor amounts of fine-grained sphalerite report to the tailings.

UNLIBERATED GRAINS

In the chapter on MINERAL LIBERATIONS it was determined by material balance calculation that 72.0, 13.2 and 14.8 % of the unliberated sphalerite are recovered in the Zn-concentrate, Cu-concentrate and tailings, respectively (See Table 25). As approximately 80 % of these unliberated grains occur as sub-free sphalerite, their flotation behaviour is expected to be dominated by grains that should behave as free grains, and whose behaviour may therefore be partly modified in accordance with grain size. However, it has also been established that subordinate chalcopyrite locked with sub-free sphalerite has a significant affect on the latter's flotation behaviour (See MINERAL LIBERATIONS). Thus, material balance modelling of the behaviour of unliberated sphalerite grains through the circuit on the basis of grain size was not performed because of the expected superposition of the affects of liberation.

The grain size distributions of unliberated sphalerite are given in Table 54, and the corresponding curves for the principal products of interest are plotted in Figure 29. As in the case of free sphalerite, the grain size distribution of unliberated sphalerite in the Cu-concentrate (i.e. 80 % - 19 um) is finer than that of any of the other products of interest. Furthermore, that of the tailings (i.e. 80 % - 28 um) is also finer than that of these other products, indicating that the losses of unliberated sphalerite are principally in the finer grain size classes. The grain size distribution of unliberated sphalerite in the Zn-concentrate (i.e. 80 % - 44 um) lies slightly on the coarser side of that in the feed (except close to the 80 % passing percentile), confirming that coarse-grained unliberated sphalerite is recovered in the Zn-concentrate. By contrast, the grain size distribution of unliberated sphalerite in the Cu-scavenger concentrate is 80 % - 46 um, suggesting losses of coarse-grained unliberated sphalerite to the Cu-circuit. However, this coarse-grained fraction is largely rejected by Cu-roughing and Cu-cleaning and eventually enters the Zn-circuit, as grains of unliberated sphalerite coarser than 37 um were not observed during image analysis of the Cu-concentrate.

The grain size distributions of sub-free sphalerite are given in Table 55, while the material balance calculated recoveries of the grain size classes are given in Table 56. In MINERAL LIBERATIONS it was determined that 80.3, 12.1 and 7.55 % of the sub-free sphalerite are recovered in the Zn-concentrate, Cu-concentrate and tailings, respectively (See Tables 33 and 56). Thus, 9.30, 12.1 and 13.2 % of the free, sub-free and unliberated sphalerite, respectively are lost to the Cu-concentrate, suggesting that the degree of liberation of sphalerite has a significant, but subordinate affect on the losses of sphalerite to the Cu-concentrate. The increased losses of sub-free relative to free sphalerite indicate that the accompanying subordinate locked mineral, principally chalcopyrite, does exert an influence on the flotation behaviour of the constituent particles.

The recovery curves for the grain size classes of sub-free sphalerite in the Zn-concentrate, Cu-concentrate and tailings are given in Figure 30. While increasing losses of free sphalerite with decreasing grain size is observed in Figure 28, the pattern is signiicantly more pronounced for sub-free sphalerite. This is partly due to losses to the Cu-concentrate, and also to

the tailings, as calculated by MATBAL. However, the latter losses of the finer classes of sphalerite to the tailings are not apparent from the raw image analysis data (See Table 55), nor from petrographic observation. The losses of 49.4 and 20.6 % of the - 4.60 and - 6.50 um classes, respectively to the tailings are anomalous, and in view of the uncertainty in the raw data are most probably erroneous. In addition, these apparent losses would tend to force the grain size distribution of unliberated sphalerite in the tailings to the finer side of that in the Cu-concentrate, in contrast to that of Figure 29 and Table 55. Furthermore, as approximately only 2 % of the sub-free sphalerite occurs in the - 4.60 and - 6.50 um classes combined, these apparent losses are minor in terms of the modal fraction of Zn in the feed.

The most significant feature in Figure 30 is the high recoveries (i.e. 30 to 45 %) in the Cu-concentrate of the - 9.2, - 13.0 um - 18.4 and - 26.0 um classes of sub-free sphalerite. These classes comprise 20 to 30 % of the unliberated sphalerite in the feed, are characterised by abundant observations and hence favourable counting statistics, and yield apparently good material balances i.e. minor adjustments to the raw data. The high recoveries in these classes, falling off sharply above the - 26.0 um class, are in good agreement with both petrographic and image analysis which indicate a median grain size of sub-free sphalerite in the Cu-concentrate of approximately 10 to 15 um. Thus, the losses of sub-free sphalerite to the Cu-concentrate do not correspond to the slime or over-sized fractions, but rather, as in the case of free sphalerite, principally to to the 10 to 25 um range; a range nominally expected to be rejected during Cu-flotation. In fact, 75, 78 and 79 %, of the free, unliberated and sub-free sphalerite, respectively in the Cu-concentrate occur in the + 6.50, - 13.0, - 18.4 and - 26.0 um classes combined (See Tables 52, 54 and 55). However, it is noted that rare grains of free sphalerite up to 150 um in diameter were petrographically observed in the Cu-concentrate.

GALENA

The raw grain size distributions of free and unliberated galena in the 17 mill products are given in Tables 57 and 58, respectively. Material balance calculated modelling of the behaviour of galena on the basis of grain size classes was not performed because of the generally poor counting statistics for individual size classes of galena.

FREE GRAINS

It was determined in MINERAL LIBERATIONS that 7.69 % of the galena occurs as free grains, and of which 37.2, 41.2 and 21.7 % is recovered in the Cu-concentrate, Zn-concentrate and tailings, respectively (See Tables 20 and 21). It was also interpreted that while the behaviour of free galena is obscured by a stereological bias, significant losses to the tailings do occur. The grain size distribution of free galena in the feed is 80 % - 9.5 um, and is largely coarser (except above the 70 % percentile) than that of the Cu- and Zn-concentrates (See Table 57 and Figure 31). The approximately equivalent recoveries of free galena of equivalent grain size into the Cu- and Zn-concentrates, indicates that galena is only marginally floatable using Minerec, and that it's recovery is consequently somewhat random. In contrast,

TABLE 52

GRAIN SIZE DISTRIBUTIONS OF FREE SPHALERITE IN THE 17 MILL PRODUCTS.

	Product.	Classes (um). -3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	Area, (um^2)
1	Float Feed.	0.05	1.03	9.73	18.5	20.9	20.0	10.4	6.05	13.3	0.00	0.00	0.00	0.00	20900.
2	Cu Scav Co.	0.34	2.00	4.96	8.61	11.8	16.2	16.4	27.5	5.83	6.38	0.00	0.00	0.00	33400.
3	Cu Roug Co.	0.12	0.95	3.54	8.37	15.0	19.2	14.3	15.5	9.60	10.9	2.63	0.00	0.00	164600.
4	Cu Cl 1 Fd.	0.01	0.86	6.58	11.8	16.4	22.3	18.6	12.2	11.3	0.00	0.00	0.00	0.00	76100.
5	Cu Cl 1 Co.	1.30	4.25	9.42	13.4	20.8	19.1	14.4	17.3	0.00	0.00	0.00	0.00	0.00	51200.
6	Cu Cl 2 Co.	1.13	5.01	11.1	16.0	19.6	19.5	19.6	8.18	0.00	0.00	0.00	0.00	0.00	41500.
7	Zn Roug Fd.	1.71	4.32	7.68	13.2	17.5	16.0	9.96	4.40	7.61	17.7	0.00	0.00	0.00	14700.
8	Zn Roug Co.	0.48	1.69	4.02	6.75	9.66	13.2	16.1	18.6	14.3	10.4	4.78	0.00	0.00	2050000.
9	Zn Cl 1 Co.	0.11	1.30	4.06	6.91	10.3	13.7	15.7	18.2	17.0	7.26	5.59	0.00	0.00	742000.
10	Zn Cl 2 Co.	3.08	3.85	6.00	8.19	12.5	14.2	17.4	15.6	13.4	5.78	0.00	0.00	0.00	359000.
11	Cu Roug Tl.	0.00	0.00	0.00	2.49	6.62	18.6	12.5	13.9	13.3	32.6	0.00	0.00	0.00	27500.
12	Cu Scav Tl.	0.00	0.19	1.41	4.71	7.96	17.5	18.6	10.9	29.0	4.52	5.19	0.00	0.00	96700.
13	Cu Cl 1 Tl.	0.09	1.01	3.87	8.39	11.6	15.8	21.2	19.4	11.2	7.64	0.00	0.00	0.00	164900.
14	Cu Cl 2 Tl.	0.81	4.87	11.2	16.2	20.3	18.4	14.4	9.79	4.08	0.00	0.00	0.00	0.00	72100.
15	Zn Cl 1 Tl.	0.96	6.84	18.4	17.2	11.8	13.0	13.2	18.6	0.00	0.00	0.00	0.00	0.00	4380.
16	Zn Cl 2 Tl.	0.46	4.95	7.29	10.6	10.7	12.2	8.56	29.3	16.0	0.00	0.00	0.00	0.00	13000.
17	Zn Roug Tl.	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.

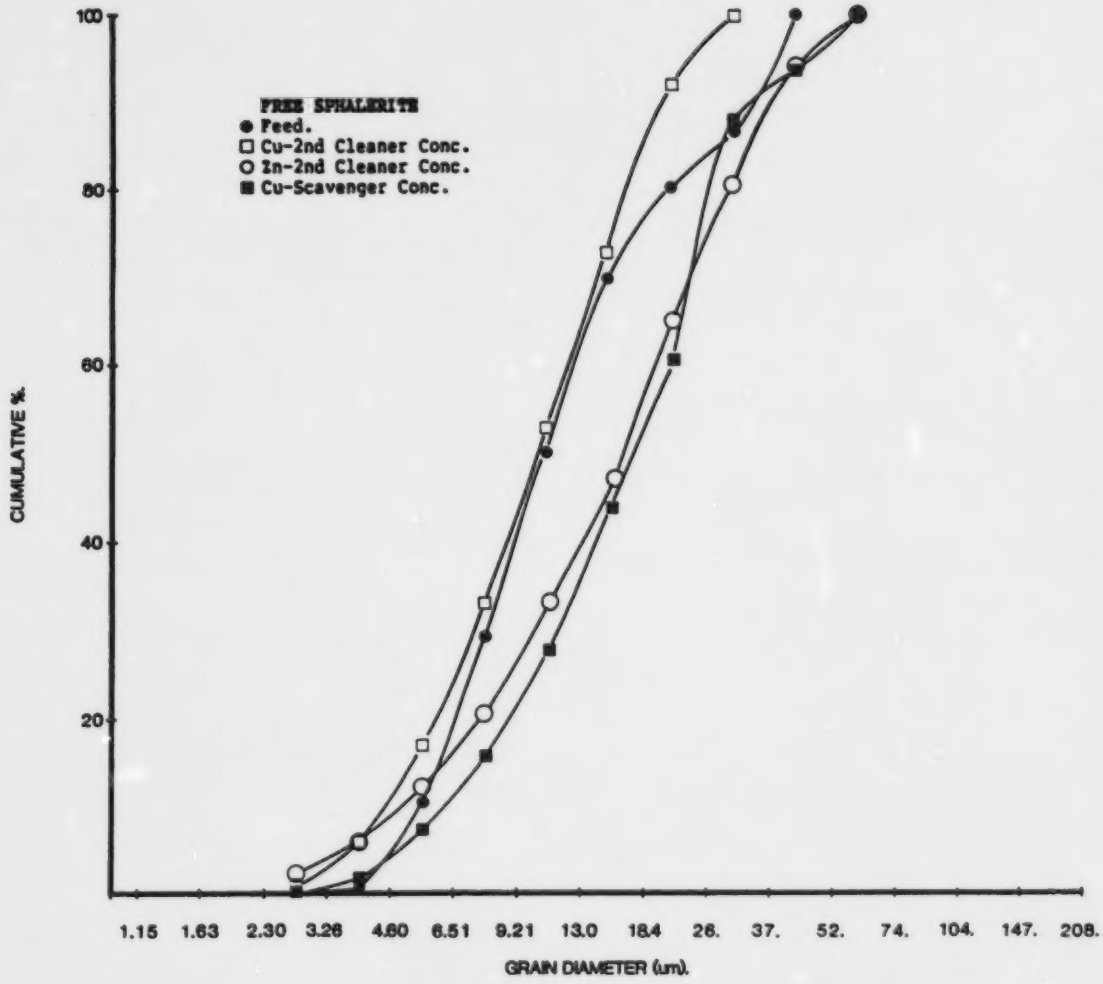


FIGURE 26.
THE GRAIN SIZE CURVES FOR FREE SPHALERITE IN SELECTED MILL PRODUCTS.

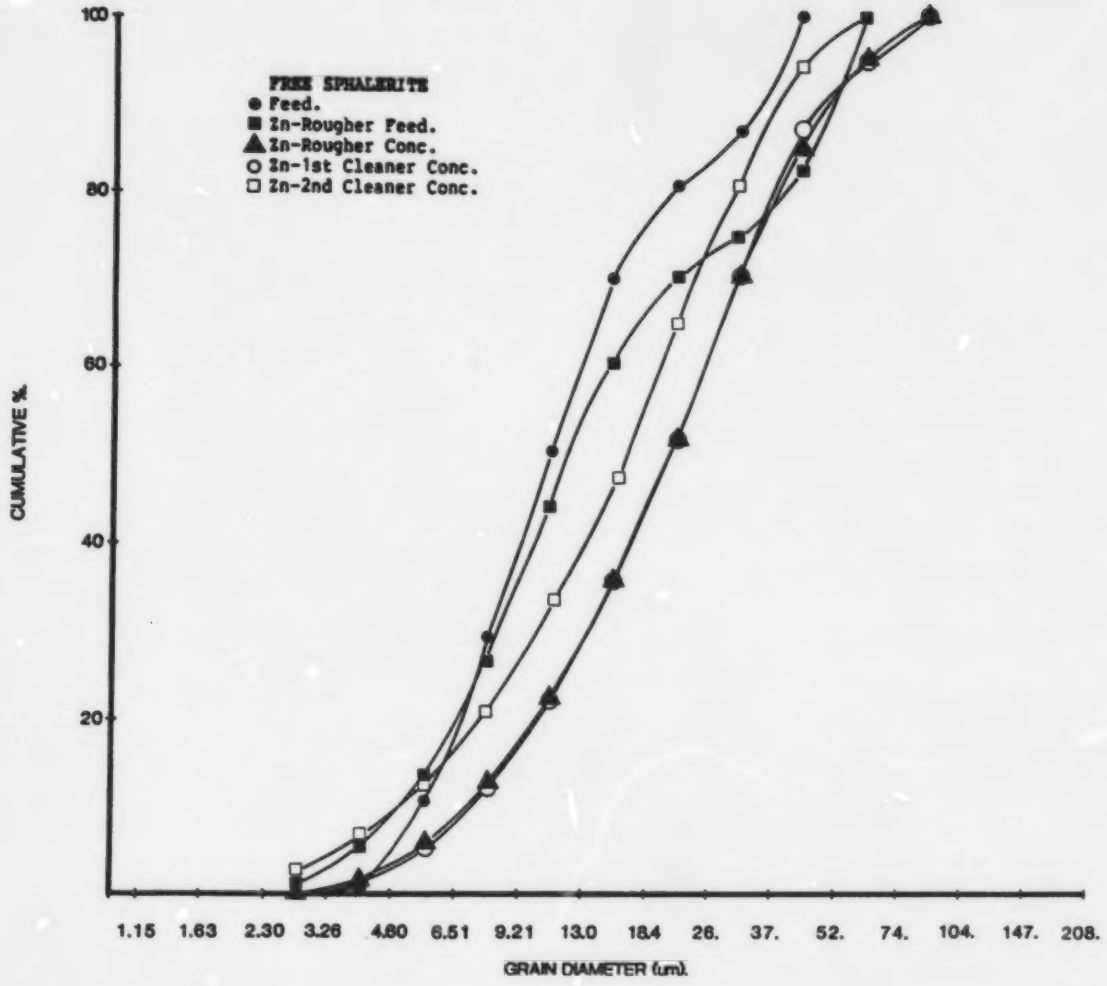


FIGURE 27.
THE GRAIN SIZE CURVES FOR FREE SPHALERITE IN SELECTED MILL PRODUCTS.

TABLE 53

RECOVERY OF THE GRAIN SIZE CLASSES OF FREE SPHALERITE.

	Product.	TOTAL	Classes (um).							
			-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0
1	Float Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2	Cu Scav Co.	15.8	5.19	2.31	3.75	2.15	2.41	3.23	9.28	1.26	2.48
3	Cu Roug Co.	35.9	19.6	19.2	40.3	23.7	23.3	22.3	39.7	14.2	0.94
4	Cu Cl 1 Fd.	38.3	32.2	28.2	52.7	29.8	27.9	28.0	45.3	14.2	0.99
5	Cu Cl 1 Co.	15.6	22.1	21.8	36.9	20.5	16.6	5.93	17.5	0.06	0.12
6	Cu Cl 2 Co.	13.2	9.85	12.7	24.5	14.4	12.0	0.23	11.8	0.05	0.07
7	Zn Roug Fd.	89.7	95.9	89.9	78.0	86.3	88.5	100.6	90.0	99.9	99.9
8	Zn Roug Co.	74.9	89.5	88.6	76.7	85.7	88.2	100.0	89.1	99.4	98.8
9	Zn Cl 1 Co.	81.5	90.4	87.7	76.8	86.1	88.5	100.1	92.4	101.2	98.9
10	Zn Cl 2 Co.	72.0	83.8	85.9	74.2	85.1	87.6	99.2	87.2	99.3	98.8
11	Cu Roug Tl.	102.6	95.3	89.6	79.2	87.8	90.4	103.0	97.5	101.2	102.4
12	Cu Scav Tl.	86.8	90.2	87.3	75.5	85.6	88.0	99.8	88.2	99.9	99.9
13	Cu Cl 1 Tl.	22.7	9.73	6.42	15.8	9.29	11.3	22.0	27.8	14.1	0.87
14	Cu Cl 2 Tl.	2.41	12.6	9.08	12.4	6.14	4.58	5.70	5.65	0.01	0.05
15	Zn Cl 1 Tl.	2.88	5.71	2.63	2.52	0.66	0.60	0.80	1.87	0.01	0.02
16	Zn Cl 2 Tl.	9.50	6.54	1.75	2.61	1.02	0.95	0.88	5.15	1.83	0.03
17	Zn Roug Tl.	14.8	6.33	1.34	1.33	0.52	0.41	0.57	0.94	0.62	1.11

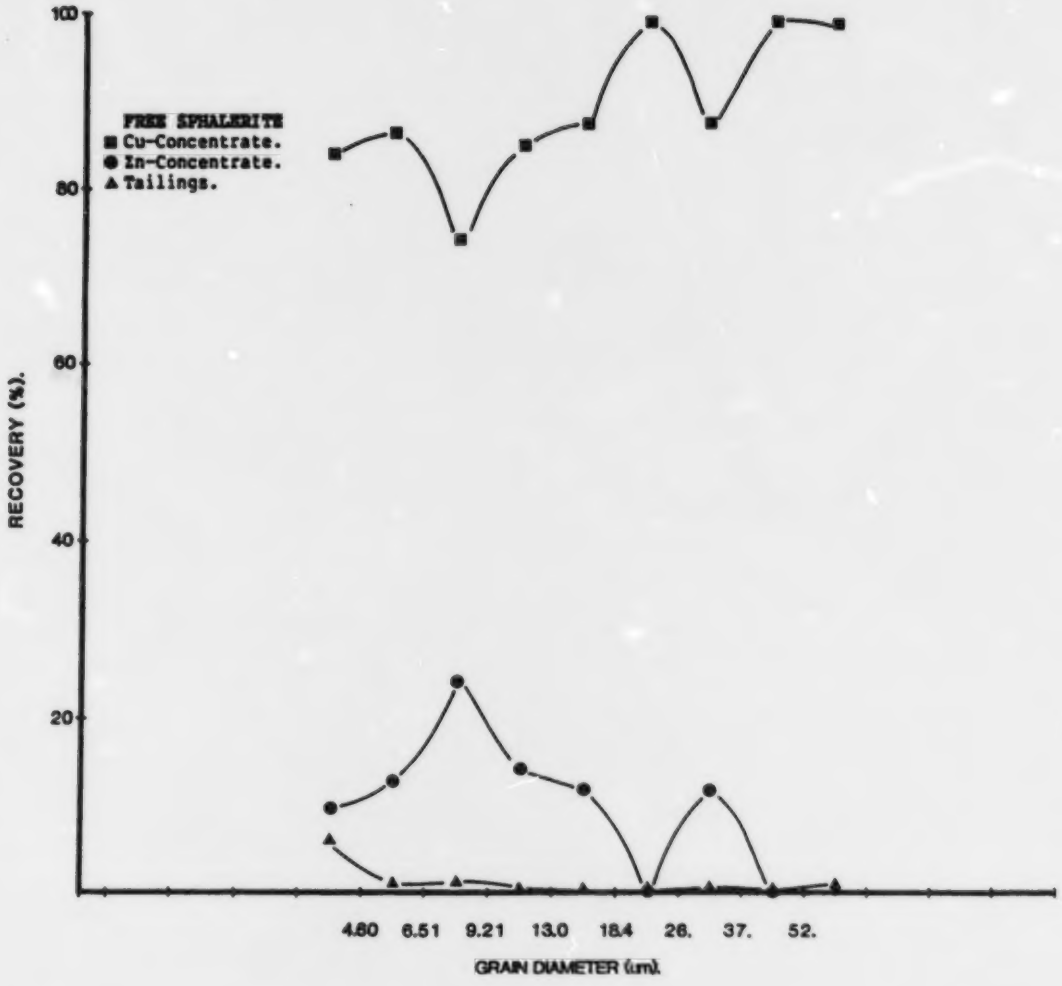


FIGURE 28.
THE RECOVERY CURVES FOR THE GRAIN SIZE CLASSES OF FREE SPHALERITE IN SELECTED MILL PRODUCTS.

TABLE 54

GRAIN SIZE DISTRIBUTIONS OF UNLIBERATED SPHALERITE IN THE 17 MILL PRODUCTS.

Product.	Classes (um).												Area (um²)
	-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	
1 Float Feed.	0.00	0.05	0.79	4.47	7.38	17.9	11.6	25.6	9.96	22.2	0.00	0.00	0.00	31700.
2 Cu Scav Co.	0.05	0.27	0.70	1.91	2.38	6.01	10.6	19.7	33.7	20.1	4.54	0.00	0.00	199000.
3 Cu Roug Co.	0.07	0.36	1.04	3.05	6.88	9.86	20.4	16.8	24.4	14.5	2.67	0.00	0.00	261200.
4 Cu Cl 1 Fd.	0.01	0.33	0.95	2.83	5.54	11.2	12.2	20.8	24.2	21.9	0.00	0.00	0.00	148000.
5 Cu Cl 1 Co.	0.39	1.33	3.74	6.49	8.37	20.4	16.1	14.5	23.1	5.55	0.00	0.00	0.00	62500.
6 Cu Cl 2 Co.	0.70	3.24	7.52	13.6	16.8	19.6	27.8	10.9	0.00	0.00	0.00	0.00	0.00	40900.
7 Zn Roug Fd.	0.89	1.34	1.29	4.50	10.6	6.86	9.54	8.54	56.4	0.00	0.00	0.00	0.00	11000.
8 Zn Roug Co.	0.09	0.29	0.69	1.47	3.15	5.61	11.9	17.3	25.4	21.2	10.4	2.64	0.00	1662200.
9 Zn Cl 1 Co.	0.03	0.24	0.72	1.36	2.64	6.29	10.8	14.5	22.2	25.4	13.1	2.69	0.00	457000.
10 Zn Cl 2 Co.	0.29	0.44	0.80	2.02	3.05	7.97	11.2	22.3	35.6	16.3	0.00	0.00	0.00	209000.
11 Cu Roug Tl.	0.05	0.00	0.08	1.24	3.61	8.93	7.32	19.3	43.6	15.9	0.00	0.00	0.00	40200.
12 Cu Scav Tl.	0.17	0.21	0.58	0.83	2.11	3.41	1.99	18.6	19.2	11.9	0.00	21.4	0.00	92900.
13 Cu Cl 1 Tl.	0.02	0.17	0.76	2.22	4.57	8.96	17.0	19.1	26.5	16.4	4.29	0.00	0.00	293000.
14 Cu Cl 2 Tl.	0.32	0.97	3.70	7.15	10.7	19.7	21.0	27.0	9.54	0.00	0.00	0.00	0.00	37500.
15 Zn Cl 1 Tl.	0.45	3.34	5.06	11.4	16.3	11.2	27.0	0.00	25.2	0.00	0.00	0.00	0.00	15800.
16 Zn Cl 2 Tl.	0.23	1.62	4.13	8.38	12.2	16.6	17.8	21.4	14.6	3.04	0.00	0.00	0.00	93100.
17 Zn Roug Tl.	0.00	1.29	2.44	7.23	24.9	20.0	0.00	44.1	0.00	0.00	0.00	0.00	0.00	2420.

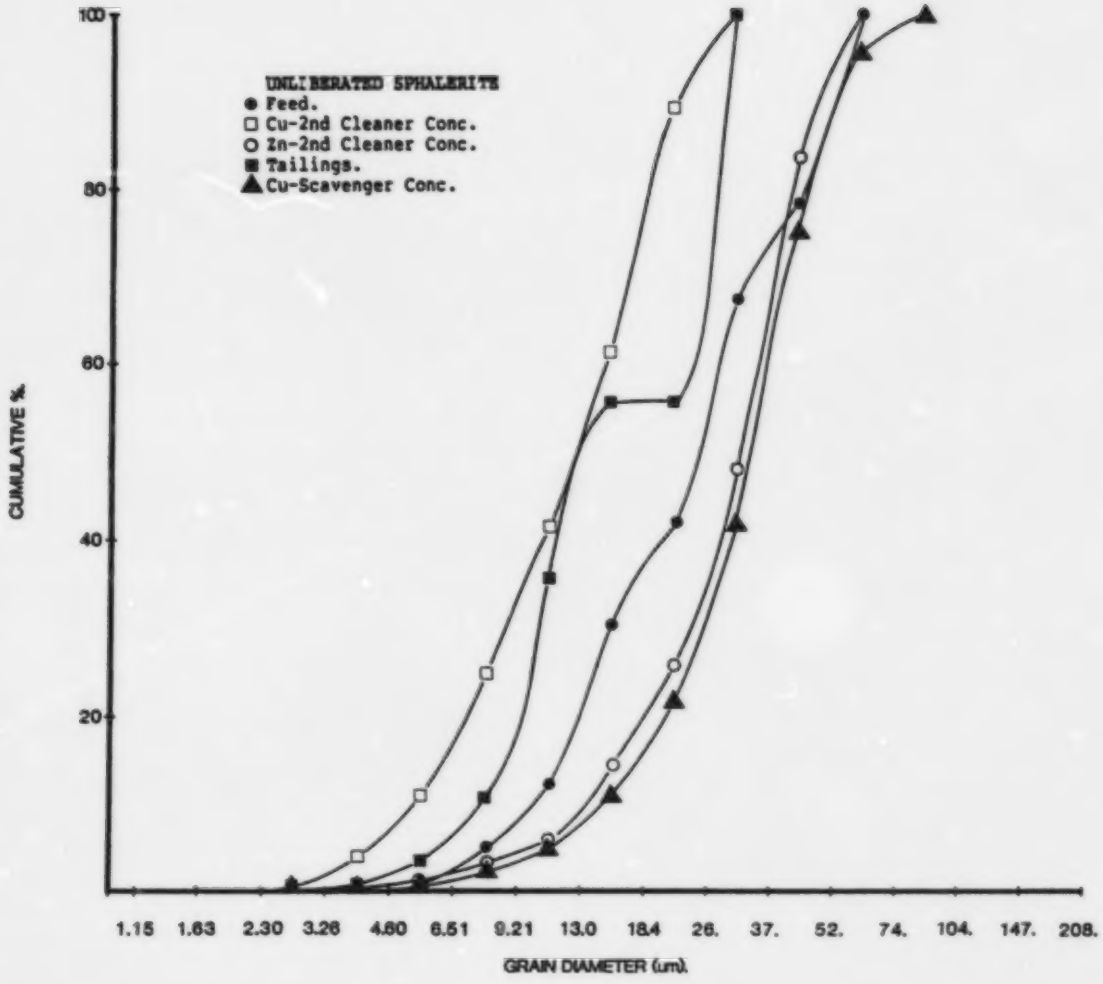


FIGURE 29.
THE GRAIN SIZE CURVES FOR UNLIBERATED SPHALERITE IN SELECTED MILL PRODUCTS.

TABLE 55

GRAIN SIZE DISTRIBUTIONS OF SUB-FREE SPHALERITE IN THE 17 MILL PRODUCTS.

	Product.	Classes (um). -3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	Area. (um^2)
1	Float Feed.	0.00	0.00	0.69	2.70	4.56	14.1	9.85	18.8	9.96	22.2	0.00	0.00	0.00	26240.
2	Cu Scav Co.	0.01	0.02	0.14	0.68	1.21	3.23	7.55	16.0	30.9	20.1	4.54	0.00	0.00	167500.
3	Cu Roug Co.	0.00	0.06	0.26	1.45	3.63	5.86	14.8	10.8	20.4	11.4	2.93	0.00	0.00	184950.
4	Cu Cl 1 Fd.	0.00	0.01	0.14	1.09	3.41	7.74	9.58	16.5	13.1	20.1	0.00	0.00	0.00	106200.
5	Cu Cl 1 Co.	0.01	0.23	0.92	2.76	6.11	14.1	13.1	12.6	23.1	5.55	0.00	0.00	0.00	48970.
6	Cu Cl 2 Co.	0.15	0.63	3.84	9.79	10.4	15.6	25.7	12.1	0.00	0.00	0.00	0.00	0.00	28890.
7	Zn Roug Fd.	0.26	0.13	0.34	1.76	7.93	5.05	4.66	8.54	56.4	0.00	0.00	0.00	0.00	9321.
8	Zn Roug Co.	0.02	0.09	0.29	0.60	1.96	3.72	8.75	14.4	23.0	20.6	10.1	2.69	0.00	1430860.
9	Zn Cl 1 Co.	0.00	0.05	0.23	0.58	0.93	3.37	7.57	12.5	19.4	23.1	13.1	2.69	0.00	381400.
10	Zn Cl 2 Co.	0.06	0.08	0.24	1.09	1.81	5.57	9.37	18.1	32.1	16.3	0.00	0.00	0.00	177590.
11	Cu Roug Tl.	0.00	0.00	0.08	0.53	2.52	6.00	6.42	17.1	39.1	15.9	0.00	0.00	0.00	35200.
12	Cu Scav Tl.	0.00	0.04	0.11	0.48	1.44	2.00	3.37	12.8	25.9	13.9	0.00	21.0	0.00	75261.
13	Cu Cl 1 Tl.	0.00	0.03	0.08	0.80	2.66	5.98	13.4	15.8	25.5	15.4	4.22	0.00	0.00	245300.
14	Cu Cl 2 Tl.	0.08	0.16	1.25	2.70	7.12	12.8	18.0	25.2	9.54	0.00	0.00	0.00	0.00	28770.
15	Zn Cl 1 Tl.	0.04	0.08	0.44	0.00	0.00	0.00	5.81	0.00	9.22	0.00	0.00	0.00	0.00	2468.
16	Zn Cl 2 Tl.	0.00	0.04	0.06	0.09	0.77	1.28	0.37	2.04	1.60	0.00	0.00	0.00	0.00	5810.
17	Zn Roug Tl.	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.

NOTE : Data presented as areal percent of unliberated sphalerite occurring as sub-free (70-99.9%) sphalerite, occurring in each grain size class.

TABLE 56

RECOVERY OF GRAIN SIZE CLASSES OF SUB-FREE SPHALERITE.

	Product.	TOTAL	Classes (um). -4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0
1	Float Feed.	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
2	Cu Scav Co.	16.4	2.36	5.99	10.4	9.38	12.0	18.9	21.6	32.8	19.2
3	Cu Roug Co.	33.5	21.7	33.1	52.2	67.9	62.8	69.7	38.4	36.1	29.9
4	Cu Cl 1 Fd.	35.8	24.4	41.5	58.2	75.7	69.8	76.0	43.8	37.6	29.9
5	Cu Cl 1 Co.	14.4	22.0	36.7	42.9	51.1	46.0	37.7	17.3	1.58	3.73
6	Cu Cl 2 Co.	12.1	19.3	28.3	36.5	43.3	38.9	31.5	12.3	0.04	3.71
7	Zn Roug Fd.	88.4	82.9	74.9	63.6	56.8	61.1	71.2	87.7	101.7	96.3
8	Zn Roug Co.	80.9	38.5	54.3	55.7	53.1	59.4	70.0	87.1	101.2	95.9
9	Zn Cl 1 Co.	81.1	38.1	52.6	56.4	56.3	62.2	68.0	88.7	100.5	95.9
10	Zn Cl 2 Co.	80.3	36.3	51.1	55.6	53.1	59.4	67.4	87.1	99.5	95.9
11	Cu Roug Tl.	104.3	83.0	77.7	73.9	66.1	73.1	87.5	109.3	132.7	115.5
12	Cu Scav Tl.	87.9	80.7	71.7	63.5	56.7	61.1	68.6	87.7	99.9	96.3
13	Cu Cl 1 Tl.	21.4	2.37	4.81	15.7	24.6	23.8	38.3	26.5	36.0	26.2
14	Cu Cl 2 Tl.	2.27	2.69	8.35	6.32	7.78	7.05	6.21	5.00	1.54	0.02
15	Zn Cl 1 Tl.	0.55	2.26	3.27	0.16	0.07	0.04	2.61	0.01	1.70	0.01
16	Zn Cl 2 Tl.	0.75	1.87	1.55	0.81	3.29	2.77	0.55	1.61	0.97	0.01
17	Zn Roug Tl.	7.55	44.4	20.6	7.89	3.68	1.68	1.15	0.62	0.48	0.40

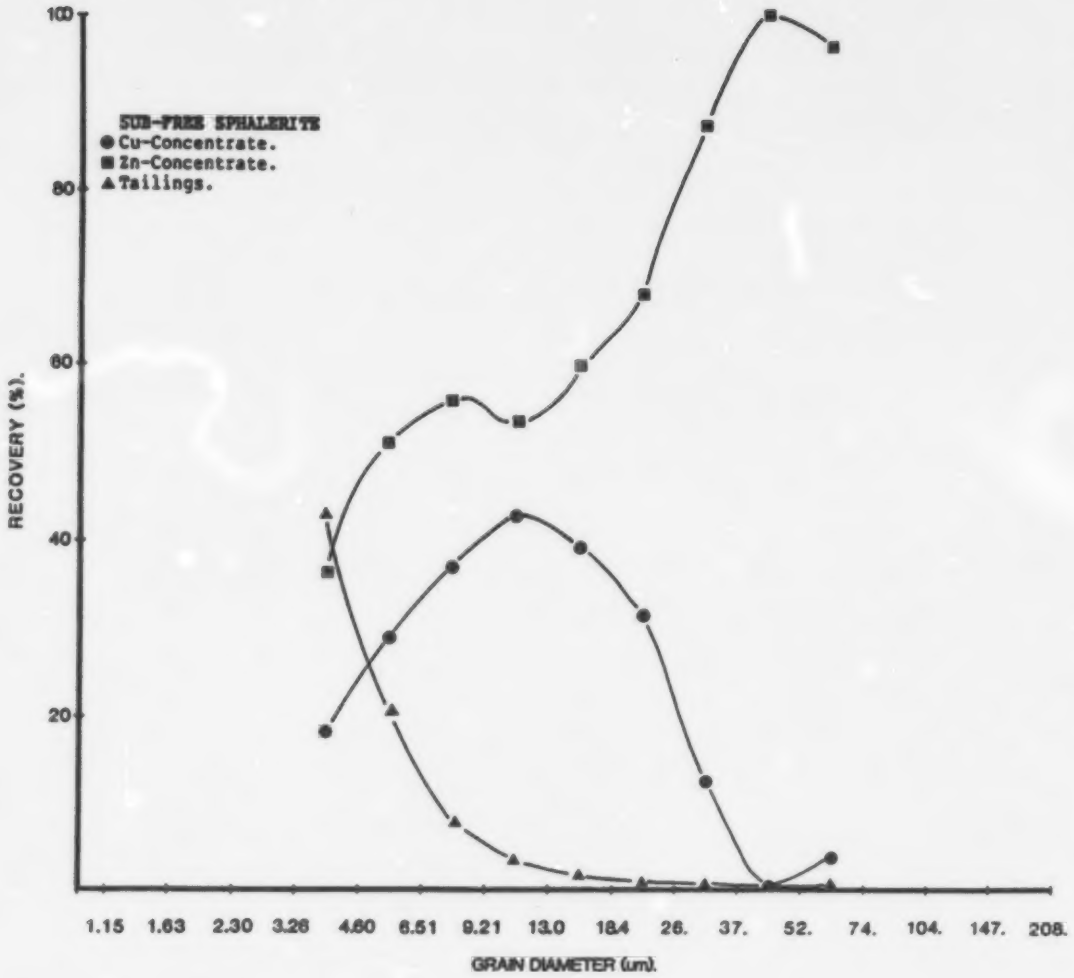


FIGURE 30.
THE RECOVERY CURVES FOR THE GRAIN SIZE CLASSES OF SUB-FREE SPHALERITE IN SELECTED MILL PRODUCTS.

the grain size distribution of free galena in the tailings is 80 % - 15 um, suggesting that with increasing grains size the high specific gravity of galena causes the particle weight to exceed the weak adhesive force with the bubble, and thus be rejected to the tailings.

UNLIBERATED GRAINS

It was determined in the chapter on MINERAL LIBERATIONS that 90.2 % of the galena occurs as unliberated grains, of which 11.2, 41.8 and 47.0 % are recovered in the Cu-concentrate, Zn-concentrate and tailings, respectively (See Tables 24 and 25). It was also determined that the classification of unliberated galena is essentially bias-free, and that the bulk of this galena occurs bimodally in the 0-10 and 90-99.9 % classes, with only minor amounts of middling grains (See Table 36). Thus, it might be expected that the behaviour of unliberated galena is dichotomous; the 0-10 class behaving in accordance with the flotation characteristics of the host mineral(s), and the 90-99.9 class behaving as free grains. However, as free grains exhibit anomalous or erratic flotation behaviour, the additional influences of the degree of liberation and the poor counting statistics prohibit any effective material balance modellling of the behaviour of unliberated galena on the basis of grain size.

The grain size distribution of galena in the feed is 80 % - 15 um, with preferential concentration of the finer size classes into the Cu-concentrate, and the coarser size classes into the Zn-concentrate and particularly the tailings (See Table 58 and Figure 30). As 89 % of the unliberated galena reports to the Zn-concentrate and tailings combined, the grain size distribution of which are comparatively coarse-grained (i.e. 80 % - 38 and - 15 um, respectively), the distribution for the feed is anomalously fine-grained. The latter is probably unrepresentative due to the poor counting statistics during image analysis of the polished section of the feed (i.e. only 5140 um^2 of unliberated galena was observed versus 9350 um^2 in the Zn-concentrate and 104300 um^2 in the tailings). The unliberated galena recovered in the Cu-concentrate consists largely of fine-grained granular and diablastic inclusions in chalopyrite, but also sphalerite-galena particles. The principal form of unliberated galena in the Zn-concentrate consists of abundant blebs and coarser grains forming inclusions in particles, with coarse-grained and generally sub-free sphalerite. It is interpreted that unliberated galena that reports to the tailings consists principally of pyrite-galena particles, where both minerals behave essentially inertly, and also sub-free and middling galena locked with sphalerite. Because of the relatively coarse-grained nature of this galena it fails to float and carries the accompanying locked sphalerite into the tailings. Thus, the losses of unliberated (and free) galena to the tailings are principally in the coarser size classes. It is noteworthy that the grain size distributions of unliberated galena in the Cu-scavenger concentrate and the Cu-1st cleaner tailings are 80 % - 31 and - 35 um, respectively (See Table 58), indicating that regrinding these combined streams may significantly reduce the losses of coarser-grained galena to the tailings.

TABLE 57

GRAIN SIZE DISTRIBUTIONS OF FREE GALENA IN THE 17 MILL PRODUCTS.

Product.	Classes (um)												Area, (um^2)
	-1.15	-1.62	-2.30	-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0	
1 Float Feed.	0.00	0.00	2.20	6.07	8.04	19.5	22.0	41.5	0.00	0.00	0.00	0.00	0.00	462.
2 Cu Scav Co.	0.00	0.00	0.00	0.00	2.66	17.3	11.1	46.1	22.7	0.00	0.00	0.00	0.00	1170.
3 Cu Roug Co.	0.00	1.43	4.19	13.1	25.0	16.3	40.1	0.00	0.00	0.00	0.00	0.00	0.00	998.
4 Cu Cl 1 Fd.	0.00	0.00	1.08	6.28	14.2	9.99	12.5	0.00	56.0	0.00	0.00	0.00	0.00	279.
5 Cu Cl 1 Co.	0.00	2.22	6.47	22.2	26.0	9.06	13.8	20.3	0.00	0.00	0.00	0.00	0.00	647.
6 Cu Cl 2 Co.	0.00	0.24	1.79	2.23	24.9	5.39	32.7	9.15	23.5	0.00	0.00	0.00	0.00	823.
7 Zn Roug Fd.	0.00	0.00	0.00	0.27	5.63	21.0	8.57	24.1	21.1	19.3	0.00	0.00	0.00	1600.
8 Zn Roug Co.	0.00	0.12	0.45	1.24	1.93	3.75	7.52	10.4	10.8	14.3	22.7	20.4	6.48	43234.
9 Zn Cl 1 Co.	0.00	0.17	1.30	1.66	3.03	6.8	9.65	13.0	33.5	30.7	0.00	0.00	0.00	5850.
10 Zn Cl 2 Co.	0.00	2.48	6.22	10.0	16.6	5.34	23.2	17.6	18.6	0.00	0.00	0.00	0.00	1360.
11 Cu Roug Tl.	0.00	0.00	0.00	0.00	1.47	5.21	6.49	5.35	8.98	20.9	51.6	0.00	0.00	3530.
12 Cu Scav Tl.	0.19	1.38	4.99	12.1	14.9	20.8	20.6	25.1	0.00	0.00	0.00	0.00	0.00	890.
13 Cu Cl 1 Tl.	0.00	0.00	0.00	0.00	0.73	3.68	9.78	23.6	17.7	14.2	5.20	0.00	25.0	11726.
13 Cu Cl 2 Tl.	0.00	0.00	0.00	0.00	1.32	9.41	16.8	19.4	13.6	6.6	22.9	0.00	0.00	5980.
15 Zn Cl 1 Tl.	0.00	0.00	0.00	0.00	0.05	0.29	2.55	7.33	5.34	14.2	29.8	12.0	28.5	31300.
16 Zn Cl 2 Tl.	0.00	0.23	0.82	1.32	2.80	4.23	7.98	14.5	26.6	23.7	9.36	8.44	0.00	25300.
17 Zn Roug Tl.	0.00	0.00	0.00	0.00	0.75	7.26	38.9	10.4	27.7	15.0	0.00	0.00	0.00	1865.

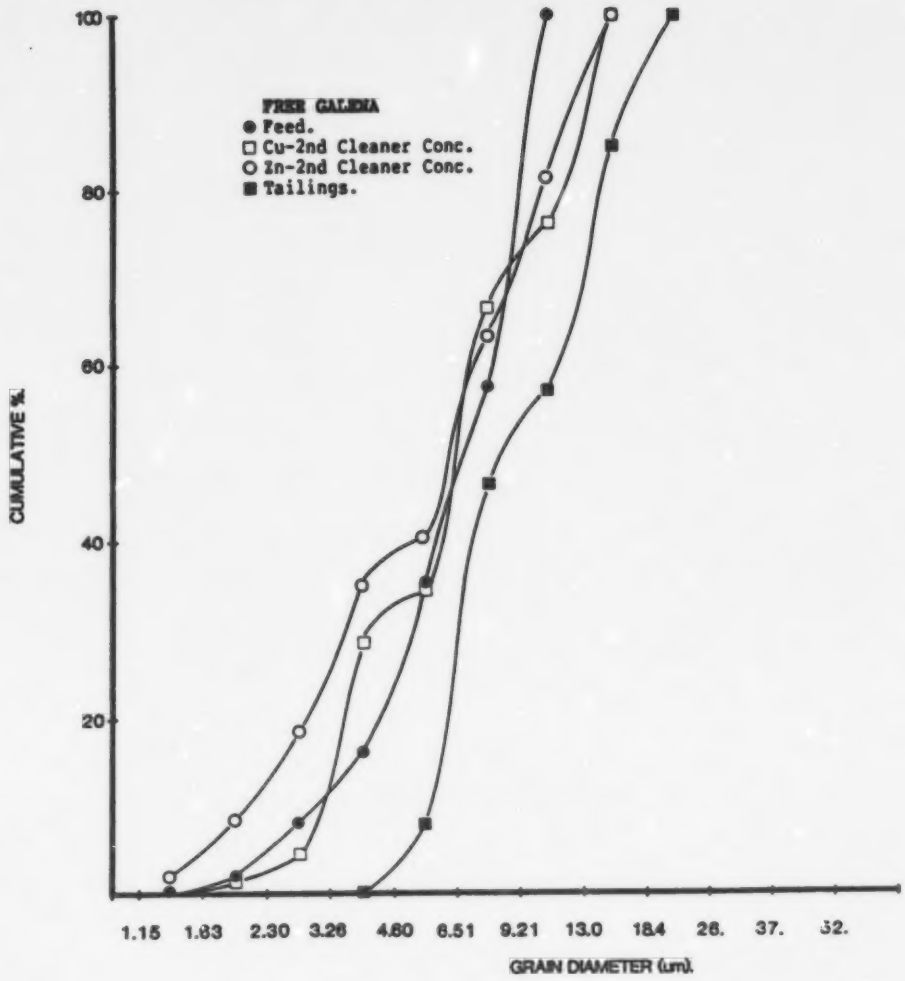


FIGURE 31.
THE GRAIN SIZE CURVES FOR FREE GALENA IN SELECTED MILL PRODUCTS.

TABLE 58

GRAIN SIZE DISTRIBUTIONS OF UNLIBERATED GALENA IN THE 17 MILL PRODUCTS.

Product.	Classes (um)												Area. (um^2)
	-1.15	-1.62	-2.30	-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0	
1 Float Feed.	0.00	0.75	2.05	4.61	4.25	9.16	19.1	18.8	22.6	18.7	0.00	0.00	0.00	5140.
2 Cu Scav Co.	0.00	0.00	0.00	0.08	1.37	3.19	6.05	13.4	24.5	16.9	14.4	20.2	0.00	12400.
3 Cu Roug Co.	0.00	0.53	3.35	7.62	19.3	14.0	17.6	18.5	5.72	13.4	0.00	0.00	0.00	3460.
4 Cu Cl 1 Fd.	0.00	0.53	3.48	9.87	22.3	14.6	18.8	30.5	0.00	0.00	0.00	0.00	0.00	712.
5 Cu Cl 1 Co.	0.00	1.09	4.63	6.94	16.1	18.6	11.5	33.7	7.65	0.00	0.00	0.00	0.00	2680.
6 Cu Cl 2 Co.	0.00	0.13	1.71	5.12	17.6	15.8	35.1	9.03	0.00	15.5	0.00	0.00	0.00	3073.
7 Zn Roug Fd.	0.00	0.00	0.00	0.23	0.95	3.47	5.70	9.39	12.2	10.6	8.56	5.52	43.7	27700.
8 Zn Roug Co.	0.00	0.09	0.26	0.61	1.10	3.20	5.75	10.3	11.2	11.8	16.2	9.33	30.1	110360.
9 Zn Cl 1 Co.	0.01	0.62	1.05	2.50	5.67	4.83	6.81	9.59	20.8	3.02	34.0	11.5	0.00	12300.
10 Zn Cl 2 Co.	0.03	0.47	1.85	4.29	5.82	5.47	12.0	8.33	11.8	14.2	0.00	35.8	0.00	9530.
11 Cu Roug Tl.	0.00	0.00	0.00	0.00	0.43	2.50	4.74	9.78	9.74	7.52	14.3	13.7	37.3	58400.
12 Cu Scav Tl.	0.02	0.28	1.45	3.57	7.09	8.82	9.84	4.85	11.6	16.1	36.4	0.00	0.00	12500.
13 Cu Cl 1 Tl.	0.00	0.00	0.00	0.00	3.62	3.78	3.75	19.1	14.3	18.8	10.9	17.3	8.58	46370.
14 Cu Cl 2 Tl.	0.00	0.00	0.00	0.03	0.77	2.19	6.52	9.32	11.6	16.2	11.1	26.5	15.8	56200.
15 Zn Cl 1 Tl.	0.00	0.00	0.00	0.00	0.00	0.15	1.86	5.09	5.45	10.4	13.8	25.8	37.4	58500.
16 Zn Cl 2 Tl.	0.00	0.10	0.25	0.87	1.82	3.10	3.58	9.36	18.0	29.6	26.0	7.45	0.00	41900.
17 Zn Roug Tl.	0.00	0.00	0.00	0.01	0.10	0.26	0.34	2.31	3.86	6.62	5.60	0.00	80.9	104300.

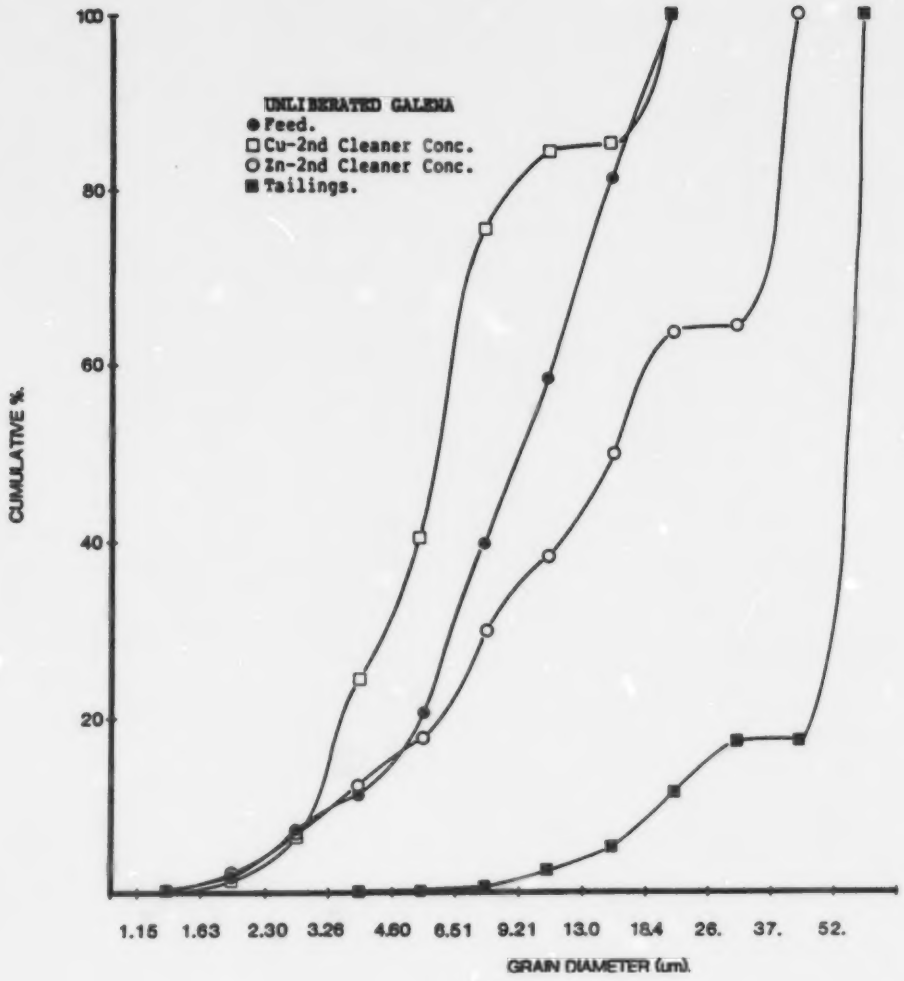


FIGURE 32.
THE GRAIN SIZE CURVES FOR UNLIBERATED GALENA IN SELECTED MILL PRODUCTS.

Chapter IX

POTENTIAL MINERALOGICAL INFLUENCES ON METAL RECOVERIES

In this chapter various aspects of the mineralogy that may affect metal recovery, and which were not directly dealt with in previous chapters, or which incorporate the findings of several analytical techniques dealt with in separate chapters, are dicussed. This chapter therefore constitutes a synthesis of the relevant data for selected mineralogical topics that warrant a more rigorous treatment.

TEXTURAL CHARACTERISTICS OF UNLIBERATED SPHALERITE IN THE CU-CONCENTRATE

Because texture deals with the relationship of mineral grains that are in mutual contact, free sphalerite grains are not treated in this section. The principal types of unliberated sphalerite in the Cu-concentrate are : (1) granular sphalerite locked principally with chalcopyrite (See Plate 1C and 1D); (2) diablastic sphalerite and chalcopyrite ± pyrrhotite (See Plate 1A and 1D); and (3) essentially free grains of sphalerite with persistent chalcopyrite-disease i.e. blebs and rods of chalcopyrite (See Plate 1B). The adjusted classification of free and unliberated sphalerite in the Cu-concentrate can be summarised as follows; 42.8 % free, 41.5 % sub-free, 8.09 % middling and 3.82 % sub-middling particles. Furthermore, because 88 % of the unliberated grains of sphalerite are locked with chalcopyrite, the classification of sphalerite locked with 'all' minerals is virtually indistinguishable from that of sphalerite locked with chalcopyrite. It is to be expected that unliberated grains of sphalerite reporting to the Cu-concentrate are locked principally with chalcopyrite. However, as 87 % of the sphalerite occurs in recoverable classes of sphalerite grains, and particularly as 43 % is absolutely free, the bulk of the sphalerite in the Cu-concentrate is recovered by a mechanism that is at least partially liberation independent. The modal fraction of the Zn recovery occurring as free and in each class of unliberated sphalerite in the Zn- and Cu-concentrates are presented as histograms in Figure 33A and B, respectively. It is apparent that the modal proportions of Zn occurring as free grains and in each class of unliberated sphalerite in the Cu-concentrate, are comparable to that in the Zn-concentrate, with a slightly higher proportion of unliberated grains, particularly middling and sub-middling grains, in the Cu-concentrate.

It is expected that if the mechanism by which sphalerite is floating into the Cu-concentrate is liberation independent, then the proportion of unliberated sphalerite locked with chalcopyrite, would be similar to that in the feed. The disparity between the proportion of the unliberated sphalerite locked with chalcopyrite in the feed (i.e. 36 %) as opposed to the Cu-concentrate (i.e. 86 %) indicates preferential concentration of sphalerite-

chalcopyrite particles irrespective of the state of liberation i.e. recoverable (70-100 classes) or non-recoverable (0-70 classes). Consequently, it is interpreted that sphalerite is marginally floatable, and that accompanying locked chalcopyrite can exert a disproportionately large influence on its flotation, that is in excess of that expected based on its volume fraction of the particle.

Chalcopyrite-disease which consists of rods and blebs of chalcopyrite oriented along rational crystallographic planes in the sphalerite is partly retained in the finely ground product (See Plate 1B). The proportion of chalcopyrite-disease in the unbroken ore that persists in the ground product is unknown, but is likely comparable to the 10 % level reported by Petruk (1985). Most of the persistent chalcopyrite-disease is encased in the sphalerite, while a smaller component occurs along the grain edges, where parting was focussed but without complete liberation of the chalcopyrite blebs. In the latter cases these chalcopyrite blebs being concentrated on the grain edges (i.e. formerly twin planes and cleavage traces) contribute disproportionately with respect to their volume, to the surface area of the particle. Therefore, as these sphalerite grains containing chalcopyrite-disease generally fall in the 90-99.9 class, their surface flotation characteristics may correspond to those of the 70-80 or 80-90 classes. Petrographic examination indicates that such grains (i.e. with persistent chalcopyrite-disease on the grain edges) are more frequent in the Cu-concentrate than in the Zn-concentrate, but nonetheless constitute less than approximately 2 or 3 % of the unliberated sphalerite in the Cu-concentrate. Thus, chalcopyrite-disease is not considered as a principal contributor to the loss of recoverable classes of sphalerite to the Cu-concentrate. In that grains of sphalerite containing chalcopyrite-disease, and which fall in the recoverable classes, are more frequent in the Cu-concentrate, merely supports the contention that this sphalerite is marginally floatable, and that the accompanying locked chalcopyrite (i.e. in this case chalcopyrite-disease) provides the requisite flotation behaviour to carry the grain above the threshold for flotation. Because less than 20 % of the unliberated sphalerite in the Cu-concentrate contains chalcopyrite-disease the mechanism of this premature sphalerite flotation cannot be ascribed to it, but merely constitutes another symptom of the 'ailment'. Furthermore, as the recoveries of essentially free (i.e. 90-99.9 class) grains of sphalerite in the Cu- and Zn-concentrates are 12.1 % and 87.8 %, respectively, and as sphalerite grains containing persistent chalcopyrite-disease are perhaps twice as common in the Cu-concentrate as opposed to the Zn-concentrate, the bulk of the persistent chalcopyrite-disease is recovered in the Zn-concentrate.

Diablastic sphalerite-chalcopyrite constiutes a major component of the unliberated sphalerite in the Cu-concentrate. Diablastic sphalerite-chalcopyrite consists of fine-grained, carious to myrmekitic intergrowths of these two minerals due to collection crystallisation during and after shearing (Healy & Petruk 1988). The lobate, interlocking nature of the intergrowths is similar to a 'jig-saw puzzle' (See Plate 1A), which because of the fine grain size is difficult to liberate on grinding. Healy & Petruk (1988) determined that the optimum grind for diablastic-textured sphalerite-chalcopyrite from the Sheared Chalcopyrite + Sphalerite ore type was 37 and 18 um, respectively, whereas at the grind of 65 % - 43 um, the predicted liberations (not reported) were 56 % and 52 %, respectively. Furthermore, Healy & Petruk

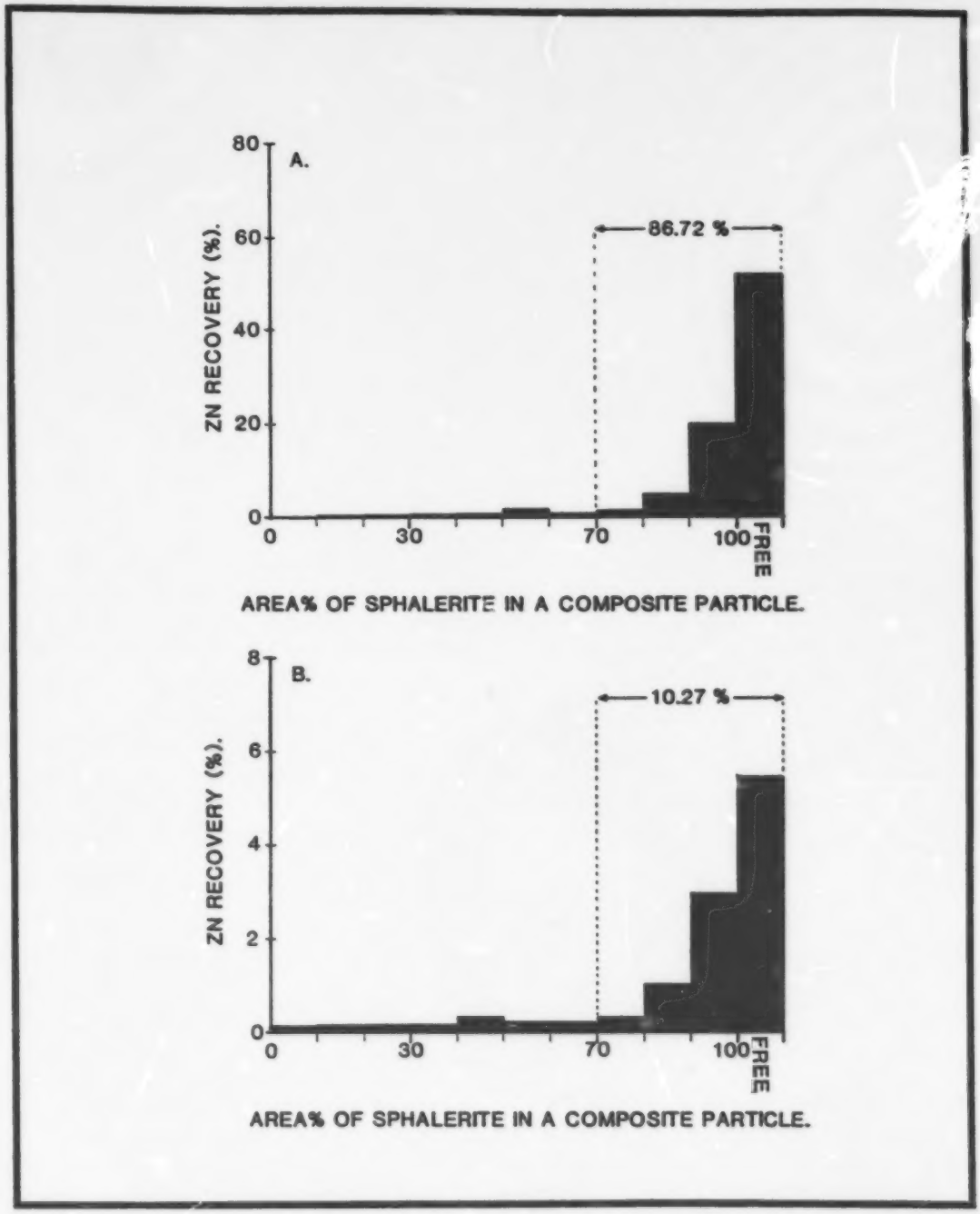


FIGURE 33.
THE MODAL FRACTION OF THE Zn-RECOVERY OCCURRING AS FREE, AND IN EACH CLASS OF UNLIBERATED SPHALERITE IN (A) THE Zn-CONCENTRATE, AND (B) THE Cu-CONCENTRATE. CORRESPONDING DATA GIVEN IN TABLE 34.

(1988) pointed out that unliberated grains derived from this texture would contain the two principal valuable minerals which would be intricately intergrown, yet destined for different products. Consequently, a liberation dependent mechanism for poor separation of Cu and Zn is envisaged for float feeds containing abundant Sheared Chalcopyrite + Sphalerite ore. Material balance calculations of the proportions of the feed contributed by each ore type indicated that only 2 % of the feed consisted of Sheared Chalcopyrite + Sphalerite ore (See ORE COMPOSITION OF THE FEED). Despite this small component of Sheared Chalcopyrite + Sphalerite ore, conspicuous diablastic-textured chalcopyrite-sphalerite particles in the Cu-concentrate, and diablastic-textured sphalerite-chalcopyrite and sphalerite-pyrrhotite particles in the Zn-concentrate, were observed petrographically (See Plate 1A, 1D, 2A and 2D). The diablastic chalcopyrite-sphalerite particles comprise a significant component (i.e. approximately 20 %) of the unliberated sphalerite in the Cu-concentrate, the latter being principally of the simple granular chalcopyrite-sphalerite type.

The persistence of a significant, but unknown proportion of the diablastic chalcopyrite-sphalerite intergrowths, through grinding is of more than purely academic interest. The No. 2 lens of the north zone, the largest and richest of the proven lenses, and which is still open at depth, is dominated by the Sheared Chalcopyrite + Sphalerite ore type and ore types (e.g. Mixed/Banded Chalcopyrite + Sphalerite) exhibiting transitions with it (Healy & Petruk 1988). Abundant Sheared Chalcopyrite + Sphalerite ore persists down to the 425 m level of this lens, and is projected to be at least discontinuously developed to depth. Ore from this lens contributed to the feed under study and most probably is the source of the diablastic chalcopyrite-sphalerite particles. Thus, where lens No 2 of the north zone comprises all or part of a given feed, significant losses of unliberated sphalerite to the Cu-concentrate are predicted. The use of selective regrinding of specific streams derived from such feeds would probably achieve improved Cu and Zn separation. The effect of a regrind on improved liberation of sphalerite and chalcopyrite is presently, and for feeds rich in diablastic-textured chalcopyrite-sphalerite, is masked (when evaluated soley on the basis of chemical assay) by the loss of approximately 10 % of the sphalerite to the Cu-concentrate in the form of free and sub-free sphalerite, and which is essentially liberation independent.

The granular chalcopyrite-sphalerite composite particles exhibit simple boundary relationships with essentially straight to curved and non-interlocking contacts. These particles account for approximately 55 - 60 % of the unliberated sphalerite grains in the Cu-concentrate, and are amenable to regrinding in that preferential breakage will readily liberate the sphalerite from the chalcopyrite. However, in that the majority of the sphalerite in these particles falls into the sub-free composite class, and therefore are expected (yet failed) to report to the Zn-concentrate, suggests that liberation of this sphalerite may have minimal effects on improved Zn losses to the Cu-concentrate.

THE MINERAL CHEMISTRY OF SPHALERITE

Healy & Petruk (1988) presented 59 microprobe analyses of sphalerite from 18 samples of Trout Lake ore. The average composition is 59.8 % Zn, 6.57 % Fe, 33.5 % S, 0.16 % Cu and 0.06 % Cu (mean total of 99.84 %). Surprisingly, the value of 0.16 % Cd in sphalerite is in good agreement with the value of 0.14 % Cd derived from bulk chemical assays of 1983 Zn-concentrates; assuming that sphalerite is the sole host of Zn and Cd. Cu was detected in only 5 of the analyses, and then significantly above the detection limit of 0.08 % Cu. The Cu values are erratic, with no detected values between the detection limit and 0.52 % Cu (the minimum value observed). This was interpreted as the product of analytical contamination by blebs and rods of chalcopyrite (i.e. chalcopyrite-disease) within the analytical volume, rather than Cu in solid solution.

CONTAMINATION OF MICROPROBE ANALYSIS : THEORY IN BRIEF

Though chemical compositions determined by microprobe are essentially grain discrete, adjacent grains or inclusions of other minerals can contaminate the analysis if they lie within the analytical volume. An electron microprobe consists of an electron gun, in which electrons are accelerated and focussed into a narrow beam (< 1 um in diameter), and which impacts with the polished surface of the sample either as a stationary or moving/rastering point. Numerous emmissions result from the interaction of the electrons with the sample, of which secondary electrons (SE), backscattered electrons (BSE) and particularly X-rays are the most significant. The nature and intensity of each type of emmission is measured by one or more dedicated detectors, and provides information on the physical and chemical nature of the sample at the point of analysis. The composition is determined by relating the intensity of characteristic X-rays for the constituent elements generated within the analytical volume to those of a standard. The analytical volume consists of a central tear-drop shaped core, which is the volume of electron interaction and primary X-ray generation. In the case of sphalerite (Z = 25) this volume is approximately 1 um^3, with maximum horizontal radii of 1 um. Thus, provided the point of analysis lies 1 um or further from foreign minerals, primary X-rays from elements of the latter are not generated. However, the operator cannot be certain that an inclusion lies below the surface of the sample, and within the analytical volume. Abundant chalcopyrite-disease makes this particularly true in the analysis of sphalerite.

In addition to primary X-rays, secondary X-rays contribute (to a lesser extent) to the observed characteristic X-rays. These are produced in an extended envelope around the central primary X-ray generation volume, and are generated not by the incident electrons, but by high energy primary X-rays (e.g. ZnKß) which travel considerable distances into the sample. Thus, a ZnKß X-ray may travel 50 um laterally through the sphalerite and excite CuKα in an adjacent bleb of chalcopyrite, and this CuKα may exit the sample and arrive at the detector and hence contribute to the observed chemical composition. The ZAF corrections routinely applied to microprobe analyses cannot satisfactorly account for fluorescence of elements in foreign objects and such boundary effects. ZnKα and ZnKß can travel significant distances through sphalerite (i.e. ZnKß is only 80 % absorbed 75 um from the point source), while only ZnKß

is energetic enough to fluoresce CuKα, but is extremely efficient at doing so (Healy & Petruk In Prep).

Scott (1983) reports that on the basis of experimental data the solid solubility of Cu in sphalerite at 400° C (the probable maximum equilibration temperature of Trout Lake sphalerites) is less than 0.1 Wt % (See also Hutchinson & Scott 1981). Thus, even minor Cu detected in sphalerites grains in the ore, and in the Cu-concentrate would be viewed as anomalously high, and interpreted to be the product of this analytical contamination. Contamination is much more probable in the Cu-concentrate, in which the analyzed sphalerite grains are fine-grained (i.e. a ground product) and contain, are locked with, are in contact with, and/or are surrounded by, grains of chalcopyrite. Nonetheless, the distinct possiblity remains that Trout Lake sphalerite contains minor Cu in solid solution.

RESULTS

Wiggins & Craig (1980) report microprobe analyses of sphalerites from nine massive sulfide deposits in Manitoba. The Cu contents vary from undetected (≤ 0.10 %) to 0.40 %, whilst the maximum value reported by these authors (i.e. 0.55 % Cu) is from the Great Gossan Lead, Virginia. Ironically these authors specifically note that only the microprobe technique is capable of avoiding contamination from blebs of chalcopyrite, and thus determine the Cu content in solid solution in sphalerite. However, it is clear that many researchers ignore the potential hazards of secondary fluorescence, which are particularly pertinent when analyzing minor to trace elements in mineral A, when these same elements are major constituent elements of the adjacent mineral B. The Cu contents of 0.52 to 0.85 % for Trout Lake sphalerites (Healy & Petruk 1988) seem to exceed the solid solubility limits of Cu in sphalerite, even allowing for uncertainty in the analyses (2 std dev = 0.30 Wt %). However, the possibility of a range in Cu in solid solution in Trout Lake sphalerite, and the further possibility of preferential flotation of the Cu-rich species into the Cu-concentrate was investigated by microprobe analysis.

The composition of free and essentially free sphalerite grains in the Trout Lake feed (47 analyses) and Cu-concentrate (52 analyses) were determined by microprobe analysis on the MAC V. Only sphalerite grains greater than approximately 25 um in diameter and containing negligible chalcopyrite, either as chalcopyrite-disease or locked grains, were selected for analysis. These 99 analyses are presented in Table 1 and 2 of appendix E. The average compositions in each product are:

	Zn	Fe	S	Cd	Cu	Total
Feed.	59.02	7.02	33.14	0.27	0.05	99.68
Cu-Conc.	59.28	6.91	33.07	0.23	0.49	99.98

The average Zn, Fe, S and Cd contents are indistinguishable (within the limits of uncertainty) between those of of the feed and the Cu-concentrate, and suggest that there is no preferential flotation of sphalerite with respect to their contents of these elements. These analyses are somewhat higher in Fe (i.e. 7.02 > 6.57) and Cd (i.e. 0.25 > 0.16) and poorer in Zn (i.e. 59.2 > 59.8) than the set of analyses done on sphalerites in the unbroken ores (Healy & Petruk 1988). This merely reflects a slightly different population of sampled sphalerites. The average Cd content of Trout Lake sphalerite reported by Healy & Petruk (1988) is 0.16 % and contrasts with an average Cd content of 0.25 % for the present analyses (n = 99). Assuming sphalerite is the sole host of the Zn and Cd in the present feed (i.e. 2.57 % Zn and 0.006 % Cd) the calculated Cd content of sphalerite is 0.23 %. This agrees favourable with the determined value of 0.25 % Cd.

The Cu content of sphalerite shows an apparently clear distinction between that of the feed and the Cu-concentrate. Of the 52 analyses of sphalerite in the Cu-concentrate, 44 gave detectable Cu values, while several of the 8 analyses that failed to detect Cu contained minute CuKα peaks discernible by eye in the EDA spectrum. In contrast, only 6 of the 47 analyses of sphalerite in the feed gave detectable Cu contents. The Cu values range from undetected (<0.08 %) to 0.56 % Cu in the feed, and from undetected to 0.94 % Cu in the Cu-concentrate. No discernible systematic variations in Cu contents from core to rim were observed, argueing against contamination of the analyses by adjacent chalcopyrite grains. Possible Cu contents due to surface activation of the sphalerite by Cu adsorption would be well below detectable concentrations, and cannot be addressed by microprobe analysis.

The elevated Cu contents of sphalerite in the Cu-concentrate are therefore attributed to one of the following:

(1) there is a natural range in the Cu in solid solution in Trout Lake sphalerite, whereby the Cu-rich species preferentially float into the Cu-concentrate prior to activation with $CuSO_4$.

(2) because of: (A) the close packing of grains and the abundance of chalcopyrite in the polished section of the Cu-concentrate, (B) the relatively fine-grained size of the sphalerite grains analyzed, and (C) the low mass absorption coefficient of sphalerite and the surrounding mounting medium to ZnKβ X-rays; secondary fluorescence of CuKα in surrounding chalcopyrite grains or chalcopyrite-disease is probably generating spurious Cu values.

While the lack of systematic core to rim variation in the Cu contents of sphalerite argues against analytical contamination, experimental work on the CuFeZnS system (Hutchinson & Scott 1981 and Scott 1983) and the ability of ZnKβ to travel extended distances and fluoresce CuKα in adjacent grains, suggests that the observed Cu values in sphalerite are anomalously high. In order to verify that the detected Cu contents of sphalerite are contaminant derived, an experiment was designed to test the extent of potential contamination (Healy & Petruk In Prep.). A sample of coarse-grained sphalerite (from the Black Angel mine) and a sample of coarse-grained chalcopyrite (from the Brenda mine) were polished to the 0.05 um stage. The polished surfaces were bonded together and the sample was repolished along a plane perpendicular to the contact. Two traverses, comprising fully

quantitative point analyses every 5 to 10 um, across the sphalerite and towards the artificial created planar contact (1-2 um thick) with chalcopyrite were done on the MAC V at the University of Manitoba. Sphalerite from the Black Angel mine was selected for this experiment because it is coarsely-recrystallised, relatively chalcopyrite-free sphalerite, with moderate Fe contents. Analyses further than 40 um from this boundary gave consistently undetected Cu, except occasionally where trace interstitial films of chalcopyrite or chalcopyrite-disease were apparent. Analyses within approximately 30 um gave Cu values which closely approach 1.0 % at approximately 2 um from the boundary. The large analytical uncertainty (± 0.30 wt % at 95 % confidence limits) due to the poor counting statistics in CuKα, and despite a counting time of 250 seconds, prohibits a more quantitative statement on the maximum distance at which contamination is significant. The observed contamination is likely less than for natural sphalerite-chalcopyrite grain boundaries, because of scattering and absorption at the relatively broad synthetic boundary (resulting from the inexact fit of the two polished surfaces). Regardless of the foregoing, it is not known whether minor Cu (e.g. 0.45 Wt %) in solid solution in sphalerite can affect the flotation behaviour of sphalerite.

Subsequently, two polished sections of the + 500 mesh fraction of the Cu-concentrate, admixed with carbon were prepared. The - 500 mesh fraction was rejected as the contained fine-grained sphalerite grains are not amenable to contaminant-free microprobe analysis, while the contained fine-grained chalcopyrite would potentially contribute further to analytical contamination. The carbon was added to the + 500 mesh fraction as a dispersive, such that coarse-grained sphalerite and chalcopyrite grains are widely separated in the mounting medium. Twenty six microprobe analyses of free and essentially free sphalerite from these sections were done on the MAC V, and the results are given in Table 3 of appendix E. The mean composition of these twenty six analyses, and those previously acquired on sphalerite from the Cu-concentrate and feed are shown below.

	Zn	Fe	S	Cd	Cu	Total
Feed.	59.02	7.02	33.14	0.27	0.05	99.68
Cu-Conc. (n=52)	59.28	6.91	33.07	0.23	0.49	99.98
Cu-Conc. (n=26)	59.60	6.77	33.23	0.20	0.35	100.20

The measured Cu contents ranged from undetected (< 0.10) to 0.78 Wt % Cu, with seven of the analyses giving undetected Cu. Notably, two of these seven Cu-free analyses were acquired on an extremely coarse-grained (i.e. approximate diameter of 150 um) free sphalerite grain, with consequently minimal opportunity for analytical contamination. The lower mean Cu content of these 26 analyses further suggests, but does not prove, that the observed Cu

content in sphalerite from the Cu-concentrate is due to analytical contamination, and not due to Cu in solid solution in sphalerite, and the latters preferential recovery in the Cu-concentrate. Alternatively, it may suggest that having minimized or eliminated the source of analytical contamination, these 26 analyses constitute accurate analyses of Cu in sphalerite. Thus, this work has failed to unequivocally characterise the composition, specifically the Cu content, of sphalerite in the Cu-concentrate. However, even if the measured Cu content constitutes Cu in solid solution in sphalerite, it is highly speculative, and unsupported by other case histories, to suggest that such trace concentrations of Cu in sphalerite influence the flotation behaviour of sphalerite.

POTENTIAL ACTIVATORS OF SPHALERITE

Subsequent to the flotation of chalcopyrite in the Cu-circuit, the feed to the Zn-circuit is conditioned with $CuSO_4$. The Cu^{2+} ions in solution are adsorbed onto the surface of the sphalerite grains and allow stable Cu-xanthates to form on these surfaces. The non-polar group of the xanthate or Minerec molecule is oriented into the pulp and renders the sphalerite hydrophobic, and hence the ability to adhere to air bubbles and be transferred to the froth. This process of Cu-activation is the standard flotation method for sphalerite in Cu-Zn base metal ores. By corrolary, the premature flotation of sphalerite in the Cu-circuit may indicate that Cu (and possibly Fe) ions dissolved in the pulp water are activating the sphalerite. Leroux et al. (1986) demonstrated that Fe^{2+} ions in solution, in the presence of dixanthogen (oxidized xanthate) and at a pH range of 9.5 to 11, can cause flotation of sphalerite without Cu-activation. Whether oxidized Minerec exhibits similar behaviour is however unknown. Nonetheless, the presence of minor, but reactive marcasite and covellite in the float feed points to the possibility of premature activation of sphalerite.

MARCASITE

Marcasite a polymorph of pyrite (i.e. same chemical composition - FeS_2) is a reactive mineral which readily alters in an oxygenated environment. Its abundance in the Trout Lake ores was not determined by image analysis, because it cannot be discriminated from pyrite on the basis of its BSE or chemical signature (Healy & Petruk 1988). A manual linear traverse grain counting procedure in reflected light was done on the polished section of the float feed (i.e. TLMP-1). This grain counting procedure was relatively repeatable, and hence precise. However, the data were corrected for grain size, and the uncertainty in this correction is relatively high, such that the accuracy is probably poor, but adequate for the present purpose. The observed modal amounts of marcasite in the feed is 0.11 Vol % or 0.17 Wt %. The marcasite is principally locked as granular aggregates with pyrite, with a smaller component of free marcasite grains. All observed grains of marcasite in the feed seemed fresh and unaltered, suggesting that the fast turnover of essentially dry ore (i.e. 1 to 3 days) minimizes oxidation of the marcasite, and its potential role in the activation of sphalerite. In addition, the low

modal abundances of marcasite i.e. less than 0.2 Wt %, also argues against it having a significant effect on sphalerite flotation (W. Petruk, pers. comm. 1987).

COVELLITE

Covellite (CuS) was not observed in the Trout Lake ores (Healy & Petruk 1988). However, this mineral was observed in the Trout Lake feed, and principally concentrated in the Cu-circuit concentrates (See Plate 3F, 3G and 3H). Covellite along with the related Cu-sulfides chalcocite and digenite are known to react in the pulp and cause activation of sphalerite when present in amounts greater than 0.01 Wt % (Petruk & Mainwaring 1985). However, as only four grains and/or granular masses were observed in the polished section of the float feed, it is improbable that the covellite is sufficiently abundant to cause sphalerite activation. Furthermore, the addition of sodium cyanide to the pulp has no affect on the flotation of sphalerite (N. McEachern, pers. comm. 1987). Cyanide desorbs the surface Cu on sphalerite and reacts with the Cu^{2+} ions in solution in the pulp water to form stable cyanide complexes, thus inhibiting activation (Wills 1985). In that sphalerite flotation is unaffected by cyanidation suggests that activation by Cu^{2+} ions in solution is not the causative agent. However, Petruk & Mainwaring (1985) report that excess NaCN must be added and free cyanide be present, while Wills (1985) reports that according to the equation below, the ratio of CN^- ions to Cu^{2+} ions must be greater than 3 : 1 for the cyanidation to be effective.

$$3CN^- + Cu^{2+} = [Cu(CN)_2] + 0.5C_2N_2$$

In Trout Lake mill products, the covellite occurs in a variety of forms, all of which suggest growth by replacement of a mineral initially tentatively identified as chalcopyrite. The most common varieties are : (1) overgrowths or rims consisting of radial aggregates on (See Plate 3F and 3G); (2) fracture-fillings in (See Plate 3G); and (3) thin, spindle-shaped, doubly-convex, sub-parallel lamellae in (See Plate 3F); the chalcopyrite-like mineral. The initial interpretation of these textures was that oxidising mine waters percolating through blast-deformed ores in stope walls was producing alteration of the chalcopyrite to covellite. However, the low reflectivity, brown- and purple-yellow colour of the phase, the ease with which this phase tarnished, the variably and weakly developed polysynthetic twinning, and the common association with gangue mineral particles exhibiting lamellar intergrowths and spinifex texture were strikingly anomalous.

A microprobe study of these phases in the Trout Lake and HBMS Cu-concentrates was undertaken to positively identify the phases, and thus assess their origin. The additional phases bornite-chalcocite solid solution and chalcocite were also observed in the HBMS Cu-concentrate. Qualitative microprobe analysis of the associated gangue mineral showed it to be a Cu- and Zn-bearing silicate i.e. fayalite. Fourteen microprobe analyses of the chalcopyrite-like phase (See Table 59) place the compositions in two distinct fields within the central portion of the CuFeS system (See Fig 34A). These compositions lie within the extensive liquid field of the CuFeS system at 1100°C, but do not correspond to any known mineral, except perhaps cubanite

(See Fig 34B). Five microprobe analyses of a bornite-like phase associated with the latter phase are given in Table 60, and are also plotted on Figure 34A and B, and similarly do not correspond to any known mineral. Both phases probably correspond to various intergrowths (on a submicroscopic scale) of S-rich species of various Cu-Fe-sulfides i.e. intermediate solid solution (Iss), bornite-chalcocite solid solution idaite and cubanite, produced during Cu-smelting. Two microprobe analyses of chalcopyrite in the Cu-concentrate that were obtained during the same analytical run are also shown, and plot exactly (within the limits of uncertainty) on the composition of pure stoichiometric chalcopyrite, and confirm that distinctly different phases other than chalcopyrite are altering to covellite. Because of poor spatial control during the analyses of these fine-grained phases, the compositions are likely partly contaminated by adjacent phases. However, for the purpose of discussion the phases will be loosely termed covellite, Iss and bornite.

Nine microprobe analyses of the covellite detected Cu, Fe and S, where Fe is consistently a minor element (See Table 59). Because covellite is fine-grained and intimately intergrown with Iss, and because of the poor spatial resolution on the MAC V, the minor Fe may reflect the virtual impossibility of contaminant-free analysis. The common observation of the characteristic intense blood-red and orange-red anisotropic colours confirms the presence of near-pure covellite (CuS). The compositions of covellite are plotted in Figure 34A and B, and define a range from covellite towards chalcocite (Cu_2S) and span the compositions of blue-remaining covellite, djurleite, digenite and anilite, and likely correspond to various solid solutions and intergrowths of these phases, and are collectively termed covellite here. Some of the analyses included under covellite correspond to chalcocite, which occurs as discrete grains, granular masses (See Plate 3A) and exsolution lamellae (See Plate 3F) in Iss and bornite. The latter are probably formed during Cu-smelting, and are not an alteration product produced during weathering of the backfilled slag (See Pignolet-Brandom et al. 1986).

Petrographic examination of polished sections of slag particles sampled from the Trout Lake and HBMS rod mill feeds, revealed the presence of Iss, bornite, chalcocite and covellite (See Plate 3A and 3B), as well as Cu- and Zn-bearing fayalite with porphyritic (See Plate 3A) and spinifex textures (See Plate 3C). These latter textures are indicative of rapid cooling/quenching. This confirms that the covellite is introduced into the feed from extraneous slag used as mine backfill, and which comprises part of the mine dilution. However as hydraulic backfill (recycled tailings) is principally used at Trout Lake, the amount of slag in the feed is relatively low. As slag is identified as the source of the covellite, then pending on the bulk composition (within the CuFeS system) of the Cu-matte entrained in the slag, the temperature of Cu-smelting (i.e. 1100°C), and the variation in cooling rates, a host of complexly related Cu-Fe-sulfides are likely to form, thus producing varying intergrowths, tarnishing rates, and susceptiblity to replacement by covellite. Also given in Table 60 are two analyses of extremely Fe-rich sphalerite that occurs as rounded blebs and irregular grains within the Iss phase (See Plate 3F), and which are therefore also derived from Cu-smelting. Furthermore, several grains of pyrrhotite forming inclusions in Iss were observed in the HBMS Cu-concentrate, and most probably correspond to the Cu-rich pyrrhotite solid solution, which extends into the CuFeS ternary system at 1100°C.

Because the slag in both the Trout Lake and HBMS circuits are heterogeneous, being comprised of products from different smelting batches that formed under differing conditions of bulk composition, smelting temperature, smelting time, partial pressure of O_2, cooling rates, etc., the phase equilibria are not experimentally constrained. A much more rigorous investigation of these Cu- and Cu-Fe-sulfides would be needed in order to make more definitive statements concerning the exact phases present and their equilibration history. However, in the context of the present study the above discussion clearly satisfies the need to identify the source of the covellite; that being Cu-smelter slag. Thus, the use of slag as backfill in stopes that are likely to be reactivated (e.g. for removal of crown pillars) and the dilution of ore by slag should be discouraged, where selective flotation of sphalerite is desirable.

Abundant Iss, bornite and covellite have been identified in the HBMS float feed and Cu-circuit concentrates (Healy & Petruk In Prep.). These phase are present in amounts greater than 0.01 Wt %, and are thus most probably causing activation of sphalerite with resultant high Zn losses to the Cu-concentrate. Consequently, it is expected that the HBMS circuit water is contaminated with respect to Cu^{2+} ions in solution. Therefore, water recycled back from the thickeners, which treats both Trout Lake and HBMS Cu-concentrates, may cause activation of sphalerite in the Trout Lake circuit. Though this was initially the configuration, the practise was quickly discontinued and fresh water has since been used in the Trout Lake circuit. However, no noticeable affect on the flotation of sphalerite was recorded at that time (N. McEachern, pers. comm. 1988). Another potential source of contamination by reactive Cu-and Cu-Fe-sulfides is the traditional practise of crushing slag (for various construction and backfill purposes) in the Trout Lake secondary cone crusher (B. Barlin, pers. comm. 1988). Though Iss and covellite may become 'smeared' on the crushing surfaces, it seems improbable that sufficient contamination could arise by this mechanism to provide a lasting affect on the flotation of sphalerite fed to the circuit over a period of perhaps several days before slag is again crushed. Nonetheless, this potential source of contamination warrants investigation.

PLATE 3.

PLATE 3.

(A) Section of a slag particle in the rod mill feed. The slag consists of coarse-grained porphyritic fayalite (Fay) in a glassy silicate matrix, and with entrained grains of the Cu-matte phases Iss and covellite. Plane Polarised Light (PPL).

(B) Section of a slag particle in the rod mill feed. The slag consists of relatively fine-grained porphyritic textured fayalite in a glassy silicate matrix (Sl), and containing irregular masses of Iss and bornite intergrowths, as well as disseminated fine-grained blebs of an unknown spinel-type phase. PPL.

(C) Section of a slag particle in the rod mill feed. The slag contains well developed spinifex texture, consisting of radial aggregates of needle-like fayalite (F) crystals in a glassy silicate matrix (S), due to rapid cooling. Note fine-grained blebs of an unidentified sulfide phase (S). PPL.

(D) Zn-ROUGHER TAILINGS : Two particles of slag (Sl) showing porphyritic and spinifex textures, and containing an unknown fine-grained spinel-type phase. These particles contribute a minor fraction of Zn and Cu lost to the tailings. PPL.

(E) Zn-ROUGHER TAILINGS : Particle of slag (Sl) containing two grains of Iss with replacement or exsolution lamellae of covellite, and inclusions of Fe-rich sphalerite. Oil Immersion (OI). Partially Crossed Nicols (PXN).

(F) Cu-CONCENTRATE : Covellite forming a rim on, and replacement or exsolution lamellae in, a large grain of Iss. Also note inclusions of Fe-rich sphalerite. OI. PPL.

(G) Cu-CONCENTRATE : Bornite containing a rim of covellite, and irregular bands of covellite possibly produced by exsolution or replacement. Note also that covellite replaces bornite along arborescent fractures. OI. PPL.

(H) Cu-CONCENTRATE : Large particle of coarse-grained covellite exhibiting finely-lamellar, cross-hatched twin lamellae, and blebs of sphalerite. OI. PPL.

PLATE 3.

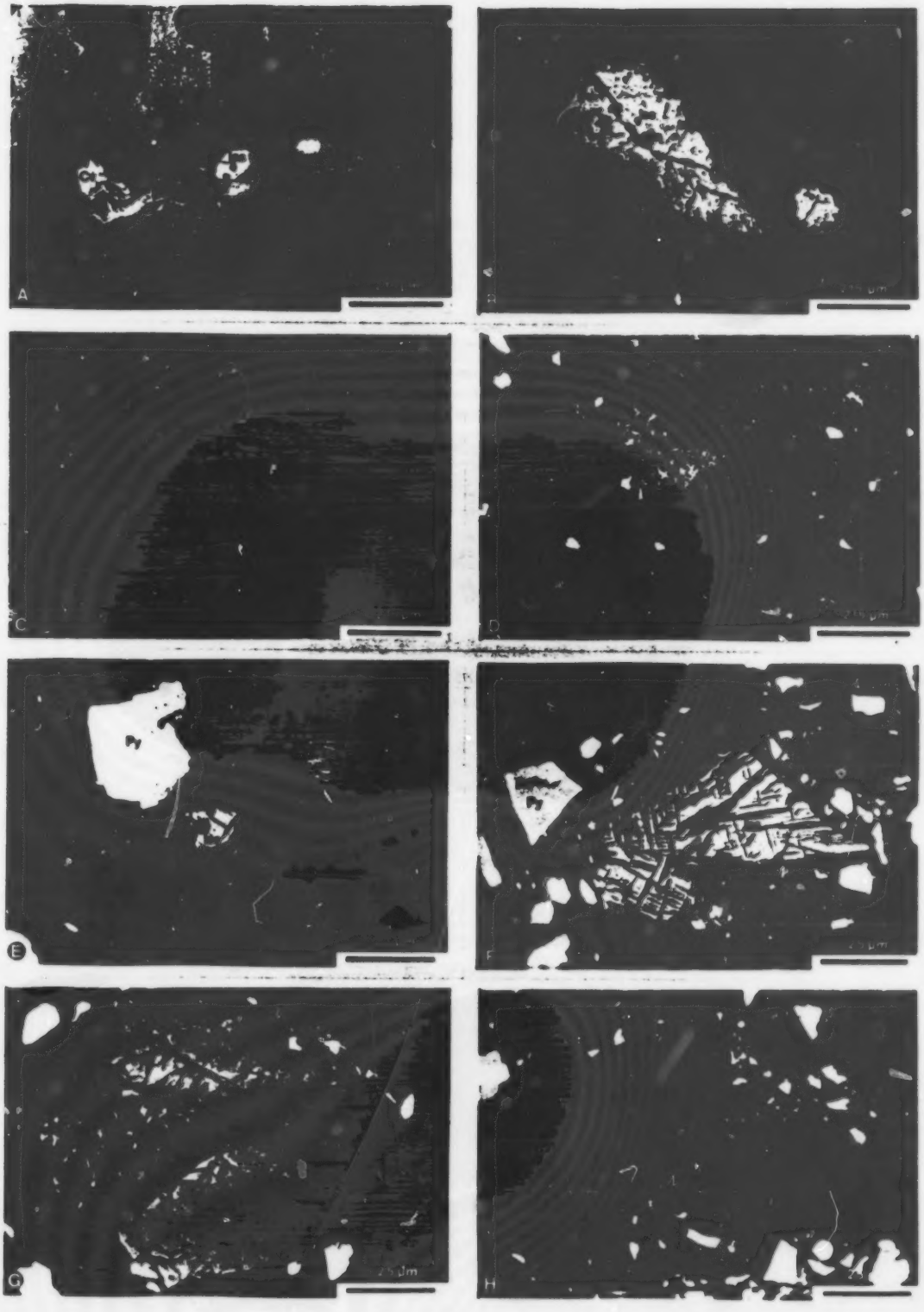


TABLE 59

ELECTRON MICROPROBE ANALYSES OF ISS AND COVELLITE IN THE TROUT LAKE AND HBMS Cu-CONCENTRATES.

	Wt %.				At %		
ANAL.	Cu.	Fe.	S.	Total.	Cu.	Fe.	S.
Iss :							
1.	48.22	20.53	31.51	100.26	35.98	17.43	46.60
2.	47.20	20.65	30.54	98.39	35.97	17.91	46.12
3.	54.80	14.99	31.50	101.33	40.83	12.70	46.48
4.	48.33	20.63	30.59	99.56	36.50	17.73	45.78
5.	47.53	21.19	33.08	101.60	34.55	17.60	47.85
6.	53.91	14.54	30.50	98.95	41.18	12.64	46.17
7.	49.78	18.84	30.85	99.47	37.61	16.20	46.19
8.	49.52	19.00	31.24	99.75	37.22	16.25	46.53
9.	47.28	20.35	31.57	99.20	35.55	17.41	47.04
10.	23.27	42.25	35.87	101.39	16.34	33.75	49.91
11.	26.35	38.81	35.68	100.85	18.66	31.27	50.07
12.	24.38	39.46	35.00	98.84	17.59	32.38	50.03
13.	24.07	39.20	35.27	98.53	17.37	32.19	50.44
14.	20.83	42.15	35.18	98.15	15.04	34.63	50.34
Covellite :							
1.	66.67	2.07	31.95	100.69	50.38	1.78	47.84
2.	69.92	1.18	27.66	98.77	55.46	1.06	43.48
3.	71.19	1.20	27.20	99.59	56.29	1.08	42.63
4.	74.66	0.59	25.79	101.04	59.05	0.53	40.42
5.	74.44	0.60	26.27	101.31	58.53	0.54	40.92
6.	69.15	0.77	30.13	100.44	53.30	0.67	46.02
7.	70.11	0.78	30.20	101.09	53.58	0.68	45.74
8.	66.00	6.22	28.71	100.93	50.78	5.44	43.77
9.	67.06	3.87	28.43	99.35	52.48	3.44	44.08

* EDS on MAC V at the University of Manitoba.
* Accelerating Voltage = 15 kV. Sample Current = 5 nA.
* Count time = 200 secs. Focussed stationary beam (1 to 2 um).
* Chalcopyrite standard for S, Fe and Cu.
* Precision (2 std. dev.) : S ± 0.25; Fe ± 0.40; Cu ± 1.15.

TABLE 60

ELECTRON MICROPROBE ANALYSES OF BORNITE (SLAG), CHALCOPYRITE (ORE) AND SPHALERITE (SLAG) IN THE TROUT LAKE AND HBMS Cu-CONCENTRATES.

	Wt %.				At %		
ANAL.	Cu.	Fe.	S.	Total.	Cu.	Fe.	S.
Bornite :							
1.	61.05	10.09	28.20	99.34	47.54	8.94	43.52
2.	48.47	18.54	31.70	98.70	36.61	15.93	47.45
3.	61.18	9.02	28.75	99.95	47.64	7.99	44.37
4.	57.39	12.95	30.12	100.46	43.54	11.18	45.28
5.	46.94	20.75	32.89	100.59	34.59	17.40	48.02
Chalcopyrite :							
1.	34.32	31.31	34.97	100.61	24.65	25.58	49.77
2.	35.04	30.46	35.42	100.92	25.05	24.77	50.18

Sphalerite :

Wt %						At %				
S.	Fe.	Cu.	Zn.	Cd.	Total.	S.	Fe.	Cu.	Zn.	Cd.
33.77	24.74	1.72	38.68	0.24	99.16	49.74	20.93	1.28	27.95	0.10
33.98	25.07	1.18	38.97	0.39	99.58	49.83	21.10	0.87	28.03	0.16

* EDS on MAC V at the University of Manitoba.
* Accellerating Voltage = 15 kV. Sample Current = 5 nA.
* Count time = 200 secs. Focussed stationary beam (1 to 2 um).
* Standards : Chalcopyrite for S, Fe and Cu in bornite and chalcopyrite, and Fe in sphalerite; Sphalerite for S, and Zn in sphalerite; and Cd-metal for Cd in sphalerite.
* Precision (2 std. dev.) : S ± 0.25; Fe ± 0.40; Cu ± 1.15 for bornite and chalcopyrite.
* Precision (2 std. dev.) : S ± 0.29; Fe ± 0.50; Cu ± 0.32; Zn ± 0.97; Cd ± 0.20 for sphalerite.

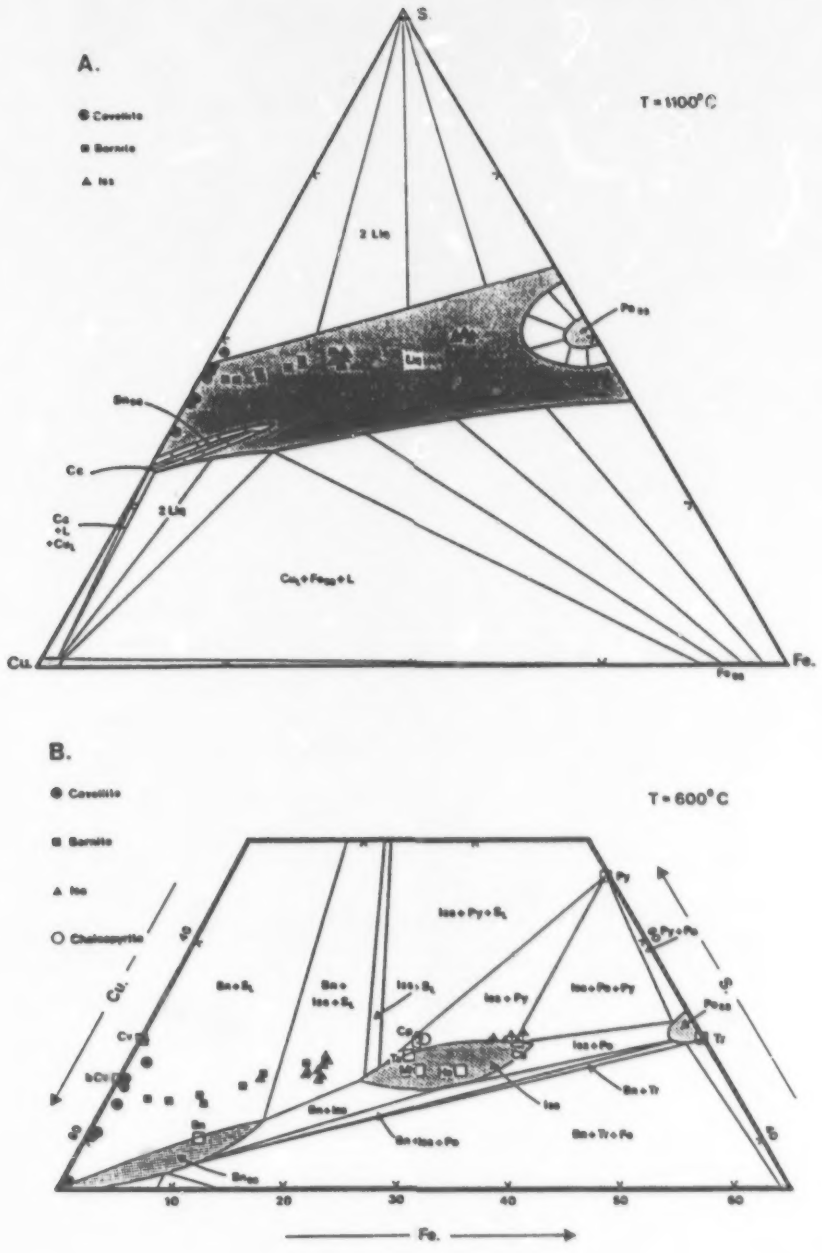


FIGURE 34.
THE PHASE RELATIONS IN THE CuFeS SYSTEM AT (A) 1100°C, AND (B) IN THE CENTRAL PORTION OF THE SYSTEM AT 600°C. THE PHASE BOUNDARIES ARE REDRAWN FROM KULLERUD ET AL. (1969) AND CABRI (1973). ABBREVIATIONS: cc - chalcocite; cv - covellite; bcv - blue remaining covellite; bn - bornite; ta - talnakhite; ha - haycockite; mh - mooihoekite; cp - chalcopyrite; cb - cubanite; py - pyrite; tr - troilite; po - pyrrhotite; Iss - intermediate solid solution; Fe - iron; subscript 'ss' - solid solution; and subscript 'l' - liquid.

THE BEHAVIOUR OF PYRITE IN THE CU-CIRCUIT

The flotation of sphalerite in the Zn-circuit is achieved by activation with $CuSO_4$, while maintenance of the pulp pH at 9.5 to 11 suppresses pyrite flotation. Consequently, pyrite is not activated or floated in the Zn-circuit, and reports to the tailings as an inert mineral. Therefore, premature activation of sphalerite would be expected to produce a considerable dichotomy in the behaviour of sphalerite and the inert pyrite in the Cu-circuit. Where activation is not the mechanism of the poor selective flotation of sphalerite, but instead is related to poor froth formation and/or collector efficiency, it might be expected that equivalent proportions of the sphalerite and pyrite in the feed may be recovered in the Cu-concentrate.

The observed behaviour of pyrite and sphalerite in the Cu-circuit is modified by their state of liberation and the degree to which each is locked with chalcopyrite. Hence, examination of the behaviour of free and sub-free grains of sphalerite and pyrite in the Cu-circuit might be expected to indicate whether activation is the causative mechanism. The liberation and classification of pyrite in these products was not determined by image analysis, except in the Cu-concentrate. While 57 % of the pyrite in the Cu-concentrate is unliberated, only 22 % of the pyrite is locked in particles with chalcopyrite, and of which 70 % occur as sub-free pyrite grains (See appendix F). Thus, only approximately 7 % of the pyrite in the Cu-concentrate can be nominally explained by it's occurrence in particles dominated by chalcopyrite, and hence pulled into the Cu-concentrate. However, it is noteworthy that much of the pyrite locked with chalcopyrite occurs as individual or clusters of pyrite grains enveloped in a veneer of chalcopyrite not liberated on grinding. In these, the pyrite will constitute the bulk of the particle (i.e. sub-free pyrite grains), yet will contribute negligibly to the surface characteristics of the particle, which should behave as essentially free chalcopyrite. Thus, perhaps as much as 20 % of the pyrite recovered in the Cu-concentrate can be accounted for by this mechanism of being 'pulled over' by chalcopyrite. Similarly, 55 % of the sphalerite in the Cu-concentrate is unliberated, but by contrast 88 % of this is locked with chalcopyrite (i.e. 48 %). It is estimated from petrographic observation that 60 to 70 % of the pyrite in the feed is free, while image analysis shows that 43 % of the pyrite in the Cu-concentrate is free. Thus, free pyrite is depleted in the Cu-circuit. In contrast, image analysis shows that 51 % and 45 % of the sphalerite in the feed and Cu-concentrate, respectively are free, indicating relatively minimal depletion of free sphalerite in the Cu-circuit. Thus, it is inferred that free sphalerite is preferentially floated, with respect to free pyrite, into the Cu-concentrate.

The material balance adjusted modal quantities of pyrite and sphalerite are 9.61 % and 3.82 %, respectively in the feed, and 8.64 % and 7.77 %, respectively in the Cu-concentrate. These values correspond to recoveries of 6.35 % and 14.37 %, respectively and concentration factors of 1.51 and 3.42, respectively in the Cu-concentrate. Thus, sphalerite is recovered in the Cu-concentrate at greater than twice the rate of pyrite, irrespective of its state of liberation. In that 43 % of the pyrite and 45 % of the sphalerite in the Cu-concentrate is free, the recovery of free sphalerite in the Cu-concentrate is approximately twice that of free pyrite. Recoveries, concentration factors and flotation efficiencies (See Tables 16, 17 and 18,

respectively) for pyrite and sphalerite indicate comparable behaviour in the Cu-rougher and Cu-scavenger, but significantly different behaviour in the Cu-1st and 2nd cleaners. Thus, initially in the Cu-rougher and Cu-scavenger, where flotation is less selective, both pyrite and sphalerite are recovered essentially in accordance with their abundance in the feed. However, in the Cu-1st and 2nd cleaners, pyrite is substantially rejected while sphalerite is rejected to a significantly lesser extent.

From Table 61 and Figure 35A it is apparent that free sphalerite is preferentially rejected with respect to locked sphalerite and pyrite (total) in the Cu-scavenger and Cu-rougher. In the Cu-scavenger particularly, the flotation efficiency of free sphalerite is very low (i.e. 2.54). At the Cu-1st cleaner, the rejection of extraneous minerals is intensified, free sphalerite exhibits the highest flotation efficiency in contrast to pyrite and locked sphalerite. This likely reflects the rejection of that initially floated/entrained proportion of unliberated sphalerite locked with gangue and pyrite, and both free and unliberated pyrite. Thus, free sphalerite and unliberated sphalerite (and perhaps pyrite) locked with chalcopyrite are interpreted to be preferentially floated. However, the breakdown of pyrite liberation in terms of the principal categories of particles containing specific accompanying locked minerals (e.g. pyrite-chalcopyrite or pyrite-sphalerite categories) is undetermined. Without quantitative data on the proportions of unliberated pyrite locked with chalcopyrite in the feed and Cu-circuit products, it is merely inferred that pyrite locked with chalcopyrite is preferentially (with respect to other liberation categories of pyrite) recovered in the Cu-concentrate.

Grains that are successfully floated at the Cu-1st cleaner are subsequently largely floated in the Cu-2nd cleaner, which thereby generates relatively high flotation efficiencies (i.e. between 66.8 and 84.6). The free sphalerite exhibits higher flotation efficiencies through Cu-cleaning (i.e. 55.5 and 70.5) than pyrite (i.e. 29.5 and 66.8). Cumulatively, the flotation efficiency through Cu-cleaning for free sphalerite (i.e. 39.1) is greater than that of locked sphalerite (i.e. 34.4) and pyrite (i.e. 19.7), suggesting the preferential recovery of free sphalerite. Thus, Cu-cleaning preferentially rejects pyrite and unliberated sphalerite locked with minerals other than chalcopyrite, and concentrates free sphalerite and unliberated sphalerite locked with chalcopyrite (i.e. 88 % of the unliberated sphalerite in the Cu-concentrate are locked with chalcopyrite). However, the unliberated sphalerite in the Cu-concentrate consists largely (76 %) of sub-free sphalerite (See Table 34), which irrespective of being locked with chalcopyrite, should report to the Zn-concentrate (See Fig 35B). Thus, while logically unliberated sphalerite recovered in the Cu-concentrate should be locked with chalcopyrite, the latter is expected to influence the recovery of middling and sub-middling sphalerite only. By corollary, only 13 % of the sphalerite in the Cu-concentrate can be explained on the basis of poor liberation and occurring in particles locked with chalcopyrite.

The behaviour of pyrite and sphalerite in the Cu-circuit is compatible with, but not unequivocally indicative of, activation as the mechanism by which sphalerite floats into the Cu-concentrate. Evidently, sphalerite is preferentially floated with respect to pyrite, and that even subordinate (i.e. comprising < 30 % of the host particle) chalcopyrite locked with sphalerite

(i.e. sub-free sphalerite) strongly influences the latter's flotation behaviour. Such flotation behaviour may indicate activation or simply poor selective flotation, which preferentially recovers sphalerite as opposed to pyrite. Wills (1985) notes that the Minerec collector is used for the selective flotation of copper sulfides from pyrite, suggesting that pyrite is very inert to Minerec, as indicated in this study. He also notes that xanthates are generally used for Cu-Zn ores, but that mineralogy permitting (not specified), the selective copper collector Minerec is used. Thus, rather than activation, which is not indicated by cyanide tests or pulp water chemistry (N. McEachern, pers. comm. 1987), but which is not incompatible with the findings of this study, it is possible that for the present mineralogy, Minerec lacks the selectivity for efficient chalcopyrite-sphalerite separation, but rejects the inert pyrite. Furthermore, the high rejection of pyrite and gangue from the Cu-circuit also suggests that the apparent lack of self-draining or 'windowing' of the air bubbles resulting possibly from the frothing properties of the Minerec collector (B. Barlin, pers. comm. 1987) is not the cause of the Zn losses to the Cu-concentrate. Whereas 14.4 % of the sphalerite in the feed is recovered in the Cu-concentrate, only 0.54 % of the gangue, 6.35 % of the pyrite and 5.99 % of the pyrrhotite are recovered in the Cu-concentrate. The low pyrite and gangue contents indicate that the Cu-concentrate is essentially 'clean', despite the high sphalerite content.

TABLE 61

FLOTATION EFFICIENCIES FOR (A) PYRITE (TOTAL), FREE AND UNLIBERATED SPHALERITE, AND (B) FREE, SUB-FREE, MIDDLING AND SUB-MIDDLING SPHALERITE IN THE Cu-CIRCUIT.

PRODUCT.	Pyrite Recovery			Free Sphalerite Recovery			Unliberated Sphalerite Recovery		
	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.
Cu Scavenger.	101.3	7.58	7.49	93.1	2.36	2.54	102.6	15.8	15.4
Cu Rougher.	130.3	29.1	22.3	113.0	19.9	17.6	138.5	35.9	25.9
Cu 1st Cleaner.	32.3	9.50	29.5	23.8	13.2	55.5	38.3	15.6	40.7
Cu 2nd Cleaner.	9.50	6.35	66.8	13.2	9.30	70.5	15.6	13.2	84.6

PRODUCT.	Sub-Free Sphalerite Recovery			Middling Sphalerite Recovery			Sub-Middling Sphalerite Recovery		
	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.	Feed.	Conc.	Effic.
Cu Scavenger.	104.3	16.4	15.7	96.0	15.5	16.2	97.3	19.8	20.4
Cu Rougher.	137.8	33.5	24.3	145.2	49.3	34.0	148.6	51.3	34.5
Cu 1st Cleaner.	35.8	14.4	40.2	53.9	24.2	44.9	53.5	24.6	46.0
Cu 2nd Cleaner.	14.4	12.1	84.0	24.2	19.5	80.6	24.6	22.5	91.5

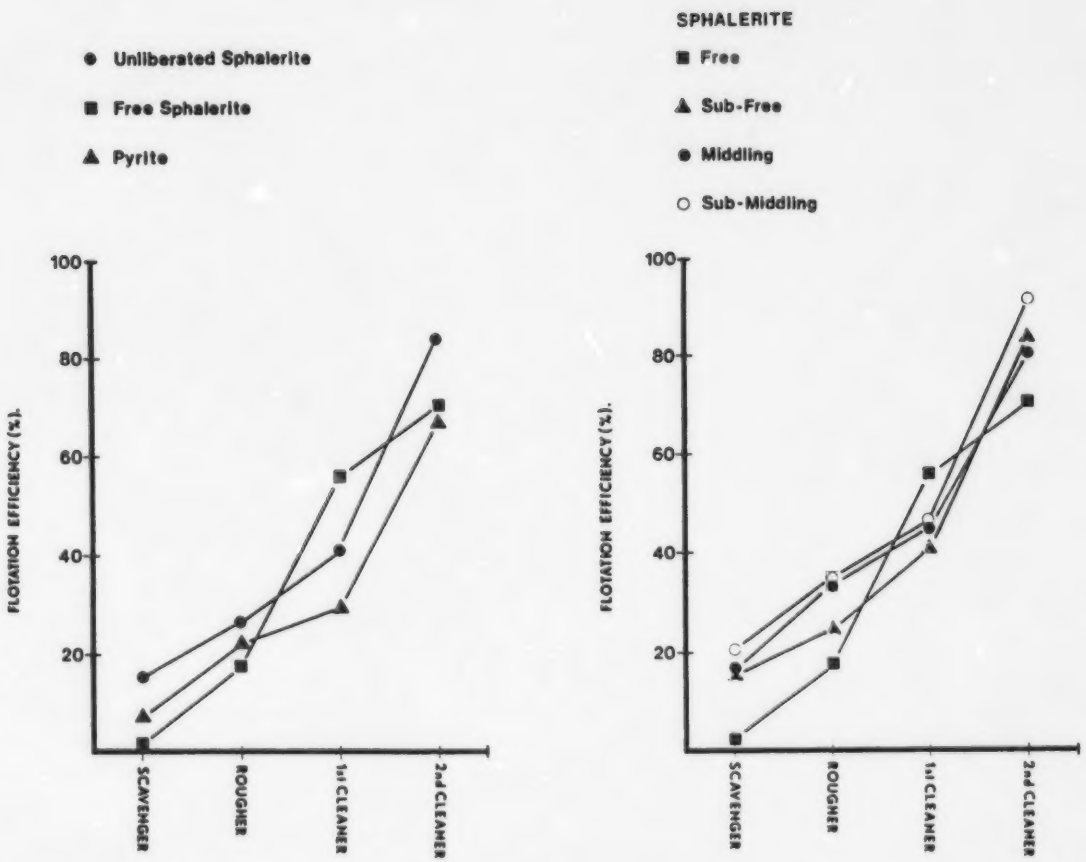


FIGURE 35.
THE FLOTATION EFFICIENCIES FOR PYRITE, FOR FREE AND FOR VARIOUS COMPOSITE CLASSES OF UNLIBERATED SPHALERITE IN THE Cu-CIRCUIT.

MINERAL-METAL ACCOUNTING

The precious metals gold and silver, and the metal cadmium are the principal by-products gained during the processing of Trout Lake ore. While the host of Cd is clearly defined as sphalerite (See THE MINERAL CHEMISTRY OF SPHALERITE, and Healy & Petruk (1988)), the host minerals of Au, and particularly Ag are not rigorously defined in terms of the numbers of minerals, and the contribution of these minerals to the total Au and Ag assays. A common misconception is that for example, a high Ag assay is completely accounted for by a petrographic examination revealing abundant freibergite and/or pyrargyrite. Healy & Petruk (1988) showed that for one sample of Trout Lake ore (e.g. TLS.5-64), the only sample with modal quantities (determined by image analysis) for the known and inferred Ag-bearing minerals, the modal amounts of such Ag-bearing sulfosalts are insufficient to account for the high Ag assay. The present study, in which the modal amounts of the Ag-bearing sulfosalts, tellurides, electrum and principal sulfide minerals, as well as chemical assays for 18 elements (chalcophilic) in each sample were determined, allows reasonable inferences on the principal hosts of Au and Ag to be made.

The trace element content of minerals are likely to vary dramatically with the nature of the mineral assemblage in a given ore sample/ore type. However, as the ore types are blended in the circuit feed, average trace element contents may be inferred. Furthermore, as the compositions of each of the streams are inter-dependent, in contrast to random samples of unbroken ore, the validity of all assumptions used are implicitly cross-checked. One basic assumption is that the precision of the chemical assays is much higher than for the modal quantities, particularly those determined by the Minor Mineral Routine e.g. freibergite and electrum (See MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES).

For a quantitative mineral-metal accounting, precise modal amounts and chemical compositions of each of the host minerals of the metal of interest are required. However, the inferred Ag content of sphalerite, chalcopyrite and pyrite and the Au content of pyrite are not detectable by electron microprobe analyses, but only by proton microprobe (micropixie) and/or secondary ion mass spectrometry (SIMS), respectively. Furthermore, while the Ag content of Trout Lake galena is above the microprobe's detection limit of 200 ppm, an error in the background correction suggests that the average value of 250 ppm Ag given by Healy & Petruk (1988) is excessively low, and more probably approaches 750 ppm Ag. The latter value is the median value of Ag in galena from the 1985 mill circuit of Brunswick Mining & Smelting (Laflamme & Cabri 1986). The composition of Trout Lake pyrargyrite and hessite are consistent and stoichiometric, while freibergite and mercurian electrum, which show wide ranges in composition (Healy & Petruk 1988 and Healy & Petruk In Prep.) are assumed to have the composition indicated by the average of the respective analyses.

SILVER MINERAL-METAL ACCOUNTING

The minerals electrum, freibergite, pyrargyrite, freieslebenite, hessite, volynskite, naumannite and acanthite contain Ag as a major constituent element. However, of these only electrum, freibergite, pyrargyrite and hessite were considered for Ag metal accounting. The remainder constitute ultra-trace minerals, whose modal quantities (being so low as to be largely un-observed during the image analysis of the mill products) were not determined. Galena (which includes the clausthalite-galena solid solution) was shown to contain Ag in solid solution by non-quantitative microprobe analysis (Healy & Petruk 1988). Various lines of evidence (Healy & Petruk 1988, and CHEMICAL ANALYSIS) suggest that the principal sulfide minerals pyrite, chalcopyrite and sphalerite contain trace, but significant Ag in solid solution (See also Petruk & Mainwaring 1985). Cabri et al. (1985) used proton microprobe analysis to directly measure the amount of several trace elements, including Ag, present in solid solution in major sulfides from three Canadian massive sulfide deposits. The most common mean values for Ag in solid solution in chalcopyrite, sphalerite, galena and pyrite seem to fall within the approximate ranges of 25 to 200 ppm, 20 to 100 ppm, 500 to 1000 ppm and 5 to 25 ppm Ag, respectively.

The contribution of each Ag-bearing mineral to the Ag assay in each product is determined by

[Modal Amounts of Mineral A (Wt %) x Ag Weight Fraction of Mineral A]

Because the Ag content of the inferred Ag-bearing minerals chalcopyrite, sphalerite, pyrite and galena is unknown, a first approximation that lies within the typical ranges (e.g. 50 ppm Ag in sphalerite) is used. These values are manually and itteratively adjusted until the Ag assay in each end product is satisfactorily accounted. Because the products represent streams with interdependent mineralogical and chemical compositions, any change in the Ag weight fraction of any mineral will induce large discrepancies in the calculated Ag assay in one or other products.

The material balance adjusted modal quantities (in Vol %) for these eight principal Ag-bearing minerals are given in Table 15, and again in Table 62A. The values were corrected for specific gravity to yield modal quantities in Wt %, and are presented in Table 62B. In the case of chalcopyrite, sphalerite and galena, the Cu, Zn and Pb assays were used as the more precise equivalents of the respective modal amounts of mineral (i.e. chemical analysis analyzes a greater and more homogeneous sample than modal/image analysis). Similarly, the Au assay was used as a modal equivalent of electrum, once having subtracted the Au assumed to occur as 'invisible' Au in pyrite (e.g. 2 ppm Au in 15.35 Wt % pyrite = 0.31 ppm Au bound in pyrite in the feed). It was found that the corrected Au assay, and the modal quantities of electrum gave comparable derived Ag assays (except in the Zn-circuit concentrates), and thus the average of the two was used. The discrepancy between the modal amounts of electrum and the Au assay in the Zn-circuit concentrates is discussed in more detail in GOLD MINERAL-METAL ACCOUNTING.

While the Ag weight fraction of freibergite, pyrargyrite, hessite and electrum are either stoichiometrically fixed, or arbitarily fixed as that of

the average composition of the mineral, the Ag weight fraction (or Ag content) of chalcopyrite, sphalerite, galena and pyrite are calculated to be 45, 55, 750 and 11 ppm Ag, respectively (See Table 63A). These values lie within the measured limits of solid solubility of Ag in these minerals (W. Petruk, pers. comm. 1988). The uncertainty in these values is undetermined, but probably high. Nonetheless, in the absence of direct analytical measurement by PIXE or SIMS, these values represent the most reasonable approximation of the true values,

It is apparent from the first row of data in Table 63A that chalcopyrite is the dominant Ag-carrier (i.e. 23 %), followed by sphalerite (i.e. 19.1 %), electrum (i.e. 18.4 %), pyrite (i.e. 13.6 %), pyrargyrite (i.e. 12.9 %), freibergite, hessite and galena. Interestingly, the four minerals freibergite, electrum, hessite and pyrargyrite which contain Ag as a major constituent element, only contribute 42 % of the Ag assay in the feed, which is exactly that contributed by chalcopyrite and sphalerite combined. Consequently, with the high recovery of chalcopyrite in the Cu-concentrate, approximately 44 % of the Ag assay in the Cu-concentrate is due to chalcopyrite. Similarly, with 90 % recovery of electrum, 25 % of the Ag assay in the Cu-concentrate is due to electrum. In addition, 4.3 % of the Ag assay in the Cu-concentrate is due to the recovery of sphalerite (i.e. 11 % recovery of Zn) in the Cu-concentrate. As sphalerite contains 19 % of the Ag in the feed, the 'sought after' improvement in Zn recovery to the Zn-concentrate will result in the loss of most of this 4.3 % Ag to the Zn-concentrate. Approximately 85 % of the Ag assay in the Zn-concentrate will then be due to sphalerite. Thus, with improved Zn-recovery, approximately 20 % of the Ag in the ore may be expected to report routinely to the Zn-concentrate.

The interpretation that approximately 42 % of the Ag in the ore is hosted in chalcopyrite and sphalerite is in good agreement with the Ag-In correlation coefficient of + 0.753 in the ores (Healy & Petruk 1988), and the similarities in the behaviour of Ag and In through the circuit (See Figure 2). The strength of the Ag-In covariance may indicate that greater than 42 % of the Ag in the ore occurs in solid solution in chalcopyrite and sphalerite. This Ag-In covariance is interpreted to reflect the coupled substitution of

In Sphalerite : $2Zn^{2+} = In^{3+} + Ag^{+}$

and possibly

In Chalcopyrite : $2(Cu^{2+},Fe^{2+}) = In^{3+} + Ag^{+}$

This substitution was observed in synthetic sphalerites (McIntyre et al. 1984), and is likely operating in chalcopyrite, which is a simple derivative structure of sphalerite.

As 14 % of the Ag in the ore is inferred to occur in pyrite, 88.8 % of which is recovered in the tailings, approximately 12 % of the Ag in the ore may be expected to report to the tailings in a form unrecoverable by flotation. By corrolary, approximately 42 % of the Ag in the tailings is bound up in pyrite (most likely in solid solution in pyrite; See Cabri et al. 1985), which is in general agreement with cyanidation tests on the tailings (Milojoković 1983 and McEachern 1987). These studies indicated somewhat higher

TABLE 62

THE MODAL QUANTITIES OF THE EIGHT PRINCIPAL Ag-BEARING MINERALS IN THE FOUR END PRODUCTS PRESENTED IN (A) VOLUME %, AND (B) WEIGHT %.

PRODUCT.	Gang	Py	Po	Cpy	Sphal	Aspy	Gal	Freib	Pyrarg	Hess	Elect	Sp. Gr.
Feed.	84.0	9.17	0.33	3.46	2.82	0.09	0.046	0.00021	0.00014	0.00004	0.00011	3.13
Cu 2nd Cl. Conc.	6.13	8.64	0.34	62.6	7.77	0.06	0.041	0.00286	0.00076	0.00044	0.00137	4.17
Zn 2nd Cl. Conc.	6.43	10.4	1.66	1.83	72.3	0.30	0.267	0.00001	0.00010	0.00001	0.00007	4.09
Zn Roug. Tail.	90.0	9.64	0.34	0.10	0.03	0.01	0.011	0.00001	0.00009	0.00001	0.00001	3.02

PRODUCT.	Cpy Cu-Assay	Sphal Zn-Assay	Gal Pb-Assay	Pyrite	Freib	Pyrarg	Hess	Electrum Modal Quantity	Electrum Au Assay	Ag Assay.
Feed.	2.156	2.572	0.038	15.35	0.00032	0.00026	0.00011	0.00053	1.92	12.4
Cu 2nd Cl. Conc.	28.00	3.992	0.070	10.36	0.00329	0.00108	0.00118	0.00493	18.5	84.8
Zn 2nd Cl. Conc.	1.745	47.29	0.381	12.74	0.00001	0.00014	0.00002	0.00026	7.33	57.9
Zn Roug. Tail.	0.117	0.191	0.018	15.96	0.00002	0.00018	0.00000	0.00005	0.33	4.16

Note : Adjusted modal quantities from Table 15; Adjusted metal assays from Table 9.

Note : Ag due to electrum calculated as mean of (A) Ag due to modal amounts of electrum, and (B) Ag = 1.13 x (Au assay - Au due to pyrite).

TABLE 63

THE (A) Ag ASSAYS DERIVED FROM MODAL QUANTITIES OF EACH Ag-BEARING MINERAL, AND (B) THE CONTRIBUTION OF THIS DERIVED Ag ASSAY TO THE ADJUSTED Ag ASSAYS (ACTUAL) IN THE FOUR END PRODUCTS.

PRODUCT. Inferred Ag content	Cpy 45 ppm	Sphal 55 ppm	Gal 750 ppm	Pyrite 11 ppm	Freib 21.7 %	Pyrarg 60 %	Hess 61 %	Electrum Modal 45 %	Electrum Assay ---	Ag Assay.
Feed.	2.85	2.37	0.33	1.69	0.69	1.60	0.67	2.28		12.5
Cu 2nd Cl. Conc.	37.1	3.67	0.61	1.14	7.14	6.50	7.20	21.5		84.8
Zn 2nd Cl. Conc.	2.31	43.5	3.32	1.40	0.02	0.85	0.12	4.71		56.2
Zn Roug. Tail.	0.16	0.18	0.61	1.76	0.02	1.10	0.18	0.30		4.31

PRODUCT.	Cpy	Sphal	Gal	Pyrite	Freib	Pyrarg	Hess	Electrum	Ag Assay.
Feed.	23.0	19.1	2.7	13.6	5.6	12.9	5.4	18.4	100.8
Cu 2nd Cl. Conc.	43.8	4.3	0.7	1.3	8.4	7.7	8.5	25.4	100.0
Zn 2nd Cl. Conc.	4.0	75.1	5.7	2.4	0.1	1.5	0.2	8.1	97.1
Zn Roug. Tail.	4.1	4.3	14.7	42.3	0.5	26.4	4.3	7.2	103.6

Ag contents bound up in pyrite i.e. 72 % Ag recovery in the pyrite concentrate from tailings, of which 21 % is recoverable; 57 % of the Ag in tailings occurs in solid solution or as sub-microscopic inclusions in pyrite. Because of the uncertainty in the present data, the value of 11 ppm Ag in pyrite is possibly an excessively low estimate. However, it may indicate that a fraction of the Ag in the pyrite concentrate not recovered by cyanidation, occurs as fine-grained inclusions of Ag-bearing minerals still unliberated or exposed after regrinding to approximately 85 % - 47 um (See McEachern 1987), and that the value of 57 % of the Ag in the tailings bound up in pyrite is excessively high.

The other major loss of Ag to the tailings (i.e. 26.4 %) is in the form of pyrargyrite; 58 % of the pyrargyrite in the feed is lost to the tailings (See Table 16). Though pyrargyrite is one third less abundant (in Vol %) than freibergite (See Tables 15 and 62A), correction for pyrargyrite's high specific gravity (i.e. 5.9), and it's high Ag weight fraction (i.e. 60 %), raises pyrargyrite to twice as significant a Ag carrier as freibergite i.e. 1.6 versus 0.69 ppm Ag contributed, respectively to the feed (See Table 63A). Interestingly, Gasparrini (1984) reports that lime, which is used in the Trout Lake circuit for pH control, is deleterious to the flotation of pyrargyrite, and that pyrargyrite is also very sensitive to changes in flotation conditions. Thus, unless the flotation conditions are significantly altered (which is not justified by the present Ag losses in the form of pyrargyrite), approximately 7.5 % of the Ag in the ore will be lost routinely as pyrargyrite in the tailings.

GOLD MINERAL-METAL ACCOUNTING

Mercurian electrum is the only Au-bearing mineral identified in the ores (Healy & Petruk 1988), and in the mill products of this study (See also Healy & Petruk (In Prep.)). Healy & Petruk (1988) observed electrum in 37 of the 112 samples that were petrographically examined, and in each of the ore types except the Vein Quartz + Chalcopyrite ore type. Considering the relatively low Au content of the ores (i.e. 1.73 ppm or 0.05 oz/t Au), and the known difficulty in locating 'gold' grains even in polished sections of gold ores, the ease with which mercurian electrum was observed suggests it is probably the principal Au-carrier in the ore. However, pyrite and arsenopyrite, both of which occur at Trout Lake, are known to contain significant 'invisible' Au in various deposit types (Wells & Mullens 1973, Boyle 1979, Hausen 1981, Springer 1983, Chryssoulis et al. 1987).

The probable mechanisms by which 'invisible' Au occurs in pyrite and arsenopyrite are: (1) solid solution; (2) interstitial lattice substituents; or (3) as fine-grained inclusions below the resolution of most available imaging systems (i.e. < 0.1 um). The specific association of Au with arsenian pyrite (Wells & Mullens 1973, Boyle 1979, Gasparrini 1983) is significant as Trout Lake pyrite is interpreted to be arsenian (See ARSENIC MINERAL-METAL ACCOUNTING). The presence of 'invisible' Au in natural pyrite, arsenopyrite and chalcopyrite at concentration levels of < 0.05 to 20 ppm has been directly measured by secondary ion mass spectrometry (SIMS) (Chryssoulis et al. 1987). In light of this, the potential hosts of Au are evaluated in terms of the

probable fractions of the Au assay carried by each mineral, and the host(s) of the high Au losses to the tailings (i.e. 25.5 %) are investigated. Through a combination of data from petrographic, image and chemical analyses, some insight into this problematic area of the Au mineralogy is achieved.

The material balance adjusted chemical assays for Au, and the recoveries of the same in each product are presented in CHEMICAL ANALYSIS (See Tables 9 and 10). Similarly, the material balance adjusted modal quantities of electrum and pyrite, and the recoveries of the same in each product are presented in MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES (See Tables 15 and 16). These data are compiled in Table 64. In Table 65, the calculated Au assay derived from the modal quantities of electrum (weight fraction of Au taken as 40 %; See Healy & Petruk (In Prep.)), and pyrite (assumed to contain 2 ppm Au) are compared to that of the adjusted chemical assay for Au. The assumption that 2 ppm Au is bound up in pyrite is derived from the fact that the tailings (i.e. sample TLMP-17) contains 17.2 Wt % pyrite and 0.65 ppm Au, and liberally assuming (as indicated by cyanidation tests) that 50 % of the Au occurs as 'invisible' Au in pyrite. Therefore, this value does not account for Au potentially bound up in arsenopyrite, is a maximum value for the 'invisible' Au content of pyrite, and implies that pyrite contains a maximum of approximately 13 % of the Au in the ore.

It is apparent that with the exception of the Zn-concentrate, the combined, inferred Au assay derived from modal quantities of electrum and pyrite in the three end products the feed, the Cu-concentrate and the tailings are in good agreement with the adjusted Au assay (actual). This appears to validate the applicability and usefulness of the material balance method in this regard, which seems to have successfully refined the raw modal quantities of electrum. The latter is fraught with sampling and analytical errors, particularly very poor counting statistics.

Major discrepancies are apparent at the Cu-scavenger, the Cu-2nd cleaner and in the Zn-circuit (See Fig. 36). While 45.3 % of the Au, only 0.9 % of the electrum reports to the Cu-scavenger concentrate. As the raw chemical assay contain much lower analytical errors, the adjustment to the Au assays, and the calculated Au recovery in the Cu-scavenger concentrate are considered more precise values. Irrespective of this discrepancy in the probable circulating load of electrum in the Cu-scavenger concentrate, the amount of electrum recovered in the Cu-rougher concentrate, and hence into the Cu-circuit proper, is in good agreement with that of the Au assay. The excess of Au derived from electrum in the Cu-concentrate over that of the Au assay (i.e. 23.0 versus 18.7 ppm Au) is well within the probable uncertainty limits of this analysis. The recovery of 9.98 % Au in the Cu-2nd cleaner tailings suggests a higher circulating load of electrum back to the Cu-1st cleaner than is apparent from the calculated recovery of electrum (i.e. 50 %). However, because of the low stream flow rate of the Cu-1st cleaner tailings, this large discrepancy in derived and actual Au assay (i.e. 11.0 versus 26.3 ppm Au), only corresponds to approximately 5 % of the electrum in the feed. No reasonable value of 'invisible' Au in pyrite and/or arsenopyrite can accommodate these discrepancies, which are most probably related to the sum of sampling and analytical errors in the modal quantities of electrum.

In the Zn-circuit the discrepancies between the Au assay derived from electrum and the adjusted Au assay (actual) are large, but relatively systematic. The assumed 2 ppm 'invisible' Au in pyrite cannot even closely accommodate the discrepancies, which are approximately half an order of magnitude. Interestingly, the Zn-circuit concentrates are the principal repository of arsenopyrite, with 45.1 % recovered in the Zn-concentrate. The three most probable explanations are: (1) the material balance analysis of modal quantities of electrum in the circuit is constrained by exceptionally poor counting statistics (i.e. precision) and is essentially erroneous, with the good agreement of the derived and actual Au assays in the Cu-circuit being simply fortuitous; (2) the electrum that is preferentially recovered in the Zn-circuit concentrates consists of fine-grained (i.e. < 1.0 um) inclusions in pyrite (which are derived from the Massive Sphalerite and the Banded Pyrite + Sphalerite ore types) that are below the spatial resolution of the image analysis routine (Petruk 1988); and/or (3) arsenopyrite is a major Au-carrier in the form of 'invisible' Au, and being principally recovered in the Zn-concentrate, is the principal host of Au in the Zn-concentrate.

While the validity of points 1 and 2 cannot be addressed, but perhaps are partly operating on the dataset, point 3 was tested by material balance calculations on the modal quantities of arsenopyrite, which are comparatively precise data. The closest balance is achieved by assuming the 'invisible' Au was essentially equally hosted in pyrite and arsenopyrite. Because arsenopyrite is almost three orders of magnitude less abundant than pyrite, the implied Au content of pyrite and arsenopyrite are 1 and 800 ppm Au, respectively. Such a partitioning of Au between a pair of co-existing minerals seems extremely high, and in as much as it fails to provide satisfactory accommodation of the Au assays, it is no more than a tenuous assertion. Nonetheless, it is concluded that pyrite and arsenopyrite are the hosts of the 10-15 % 'invisible' Au in the ore, while the exact fraction of the 'invisible' Au hosted by pyrite and arsenopyrite is unknown. However, 'invisible' Au hosted in pyrite and arsenopyrite constitutes 'refractory' Au, is unrecoverable by flotation except for those grains locked with chalcopyrite or sphalerite, and is not amenable to direct cyanidation without oxidation/destruction of these minerals.

Thus, electrum is interpreted to host approximately 85 to 90 % of the Au in the ore, of which 90 % is calculated by MATBAL to be recovered in the Cu-concentrate (i.e. 76 % of the Au). The latter value exceeds the calculated recovery of Au (i.e. 59.2 %) in the Cu-concentrate, suggesting inferior material balancing of the electrum modal quantities. This probably reflects the spatial resolution of the image analysis routine and the resulting inability to detect fine-grained inclusions of electrum preferentially locked in pyrite and arsenopyrite, and which preferentially report to the Zn-circuit concentrates. Thus, the modelled behaviour of electrum through the circuit may not incorporate that fraction of Au amenable to cyanidation which occurs as < 1 um grains. This generates discrepancies in the proportions of electrum recovered in the Zn-circuit concentrates and tailings. It may also simply reflect the poor counting statistics for the modal quantities in all the products.

In summary, the extremely large sampling and analytical errors associated with modal analysis of the trace mineral electrum, which as with other Au-

TABLE 64

MODAL QUANTITIES OF ELECTRUM AND PYRITE, GOLD ASSAYS, AND THE RECOVERIES OF EACH IN THE 17 MILL PRODUCTS.

PRODUCT.	ELECTRUM.		PYRITE.		GOLD.	
	MODAL AMOUNTS.	RECOV-ERIES.	MODAL AMOUNTS.	RECOV-ERIES.	MODAL AMOUNTS.	RECOV-ERIES.
1. Feed.	0.00011	100.0	9.61	100.0	2.23	100.0
2. Cu-Scav Co.	0.00005	0.9	37.8	7.6	52.4	54.3
3. Cu-Roug Co.	0.00120	142.2	22.0	29.1	19.6	112.0
4. Cu-Cl.1 Fd.	0.00117	147.2	22.8	32.3	20.0	121.9
5. Cu-Cl.1 Co.	0.00129	94.9	11.5	9.5	19.5	69.2
6. Cu.Cl.2 Co.	0.00137	90.0	8.64	6.4	18.7	59.2
7. Zn-Roug Fd.	0.00001	10.2	10.3	101.4	1.03	43.6
8. Zn-Roug Co.	0.00006	3.3	19.2	12.6	6.38	18.0
9. Zn-Cl.1 Co.	0.00007	4.5	26.3	18.7	9.12	27.9
10. Zn-Cl.2 Co.	0.00007	3.1	10.4	4.9	7.58	15.3
11. Cu-Roug Tl.	0.00001	10.9	10.3	101.2	2.02	86.1
12. Cu-Scav Tl.	0.00001	10.0	9.70	93.7	1.00	40.8
13. Cu-Cl.1 Tl.	0.00099	52.2	38.5	22.8	20.6	52.7
14. Cu-Cl.2 Tl.	0.00064	5.0	35.7	3.1	26.3	10.0
15. Zn-Cl.1 Tl.	0.00001	0.2	40.8	7.7	3.43	2.8
16. Zn-Cl.2 Tl.	0.00007	1.4	56.9	13.8	12.1	12.7
17. Zn-Roug Tl.	0.00001	7.0	9.64	88.8	0.65	25.5

TABLE 65

MODAL QUANTITIES OF ELECTRUM AND PYRITE, AND THE GOLD ASSAYS DERIVED FROM THESE COMPARED TO THE ADJUSTED GOLD ASSAY IN THE 17 MILL PRODUCTS.

PRODUCT.	ELECTRUM.		PYRITE.		GOLD.
	MODAL AMOUNTS. (Wt %)	DERIV-ED Au. (ppm)	MODAL AMOUNTS. (Wt %)	DERIV-ED Au. (ppm)	ADJUSTED Au ASSAY. (ppm)
1. Feed.	0.00053	2.12	15.4	0.31	2.23
2. Cu-Scav Co.	0.00019	0.76	47.3	0.95	52.4
3. Cu-Roug Co.	0.00474	19.0	29.0	0.58	19.6
4. Cu-Cl.1 Fd.	0.00474	19.0	30.8	0.62	20.0
5. Cu-Cl.1 Co.	0.00538	21.5	16.0	0.32	19.5
6. Cu.Cl.2 Co.	0.00576	23.0	10.4	0.21	18.7
7. Zn-Roug Fd.	0.00005	0.20	16.1	0.32	1.03
8. Zn-Roug Co.	0.00023	0.92	24.0	0.48	6.38
9. Zn-Cl.1 Co.	0.00027	1.08	33.7	0.67	9.12
10. Zn-Cl.2 Co.	0.00028	1.12	12.7	0.25	7.58
11. Cu-Roug Tl.	0.00005	0.20	16.1	0.32	2.02
12. Cu-Scav Tl.	0.00005	0.20	15.7	0.31	1.00
13. Cu-Cl.1 Tl.	0.00424	17.0	45.8	0.92	20.6
14. Cu-Cl.2 Tl.	0.00274	11.0	43.0	0.86	26.3
15. Zn-Cl.1 Tl.	0.00005	0.20	51.0	1.02	3.43
16. Zn-Cl.2 Tl.	0.00027	1.08	63.2	1.26	12.1
17. Zn-Roug Tl.	0.00005	0.20	17.2	0.34	0.65

minerals is characterised by the 'nugget' or 'particle sparsity' effect, prohibits a truely quantitative statement. Suffice it that Au is principally (i.e. > 85 %) hosted in electrum, and this is largely recovered in the Cu-concentrate. Cyanidation tests of the tailings indicate that approximately 50 % of the Au in the tailings (i.e. 12.5 %) occurs as 'invisible' or 'refractory' Au in pyrite and/or arsenopyrite. The fraction of the 'invisible' Au occurring in pyrite and in arsenopyrite is unknown. The exact values are a function of whether the discrepancies in the derived and actual assays in the Zn-circuit are generated through (1) a systematic bias that omits fine-grained inclusions of electrum that may be preferentially recovered in the Zn-concentrate, or (2) due to Au in arsenopyrite, which is also largely recovered in the Zn-concentrate.

Unliberated types 1 and 3 electrum (Healy & Petruk 1988) was observed in several of the product samples, though not in the tailings, despite two overnight runs (approximately 5,000 fields examined) on TLMP-17. Nonetheless, MATBAL calculated a reasonable value of 7 % losses of electrum to the tailings; this most probably occurs as unobserved type 1 electrum. A large mass of free type 2 electrum measuring 720 um^2 was observed in the Cu-concentrate, confirming that liberated electrum (mercurian) floats using Minerec, and is recovered in the Cu-concentrate (See Plate 1E and 1F). Thus, the losses of electrum are largely liberation dependent, whereas the losses of Au also include the additional effect of 'invisible' Au in pyrite and/or arsenopyrite. However, it is probable that the observed trend of increasing Au rejection with pyrite rejection in Cu-cleaning (N. McEachern, pers. comm. 1987) reflects increased rejection of coarse-grained electrum, and not simply Au as type 1 electrum or in solid solution in pyrite and/or arsenopyrite.

ARSENIC MINERAL-METAL ACCOUNTING

In view of the association of Au with arsenian pyrite in many deposits (Boyle 1979, Gasparrini 1983, Springer 1983), and because approximately 10-15 % of the Au in the ore is 'invisible' and most probably hosted in pyrite and/or arsenopyrite, the possibility of Trout Lake pyrite containing As in solid solution warrants investigation. Unfortunately, the detection limit for As(Lα) using the MAC V is approximately 0.3 Wt % (and even at these concentration levels the uncertainty is ± 0.5 Wt % As); hence As was not determined during microprobe analysis of Trout Lake pyrite (Healy & Petruk 1988). However, by material balance calculations on the abundances of arsenopyrite and As, it was hoped to infer whether the pyrite is arsenian, and to estimate the probable mean As content.

Arsenopyrite and freibergite are the only two As-bearing minerals identified in the Trout Lake ores (Healy & Petruk 1988). The trace mineral freibergite (i.e. 0.00021 Vol % in the feed) contains a very minor tennantite (i.e. As-rich end-member) component (i.e. $\bar{x}$ = 0.20 Wt % As; n = 60, Healy & Petruk 1988), and is ignored for the purpose of As metal accounting. The As assay and the modal quantities of arsenopyrite were determined for each of the 25 product samples (See Tables 8 and 14). Adjusted values and recoveries of these were calculated by material balance calculations using MATBAL and are presented in Tables 9, 10, 15 and 16, and recompiled in Table 66. It is

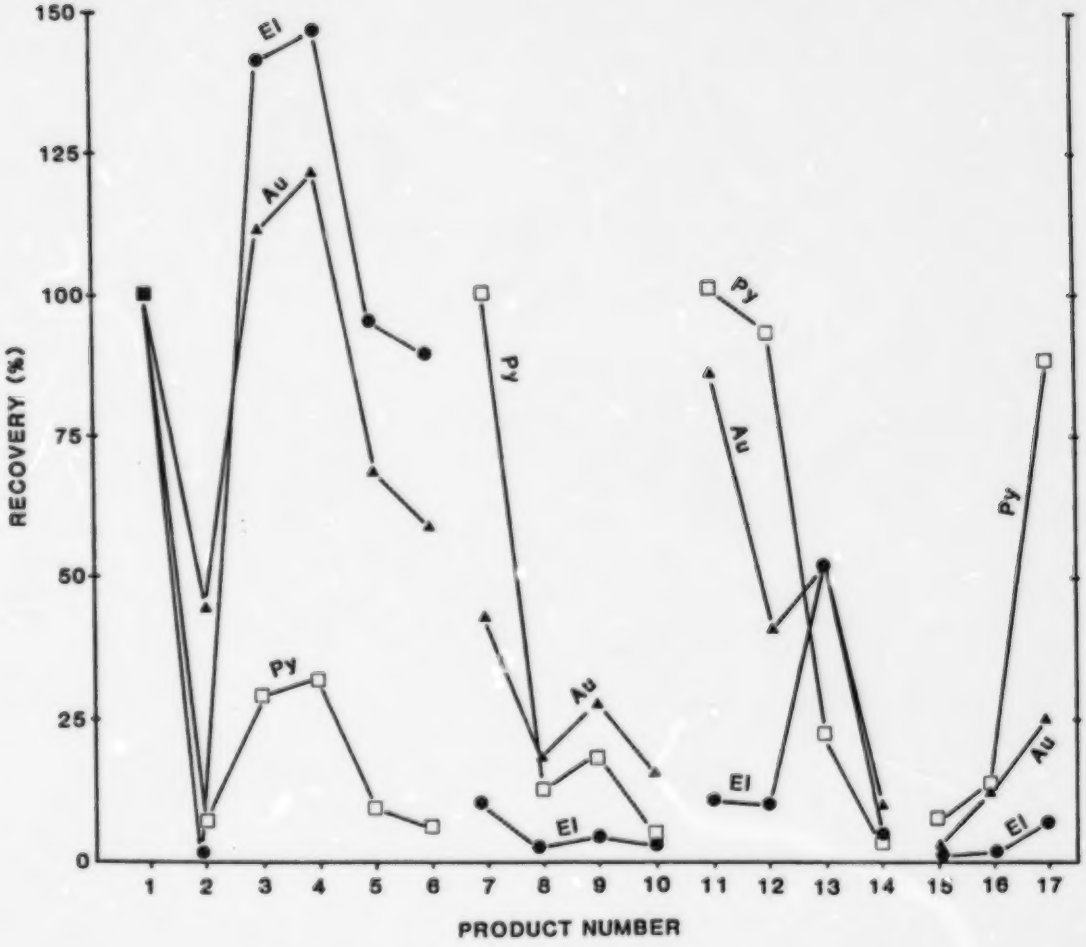


FIGURE 36.
THE RECOVERY CURVES FOR ELECTRUM, Au AND PYRITE IN THE 17 MILL PRODUCTS. PRODUCT NUMBERS AS IN FIGURE 2 AND 4. DATA IS MATERIAL BALANCE CALCULATED.

apparent from Table 66 that 3.6, 17.1 and 79.3 % of the As is recovered in the Cu-concentrate, Zn-concentrate and the tailings, respectively. In contrast however, 15.0, 45.1 and 40.0 % of the arsenopyrite is recovered in these products, respectively. The dichotomy in the behaviour of As and arsenopyrite through the circuit suggests additional significant carriers of As; most probably pyrite. The material balance adjustments to the raw modal quantities of arsenopyrite were relatively small (Compare Tables 14 and 15), and suggest that the behaviour of arsenopyrite was satisfactorily simulated by MATBAL. This stems from the relatively good counting statistics with abundant observations of arsenopyrite per sample.

The modal quantities of arsenopyrite in each product are given in Wt % Table 67. Knowing the composition of Trout Lake arsenopyrite (i.e. As weight fraction = 44.5 %; See Healy & Petruk (1988)), the amount of As contributed by arsenopyrite in each product is determined (Column labelled 'Derived As' in Table 67). With the exception of the Cu-scavenger and Cu-2nd cleaner concentrates, there is an excess of As above that which can be explained by As in arsenopyrite. The former two discrepancies likely originate from poor counting statistics for arsenopyrite in the Cu-circuit products, which are essentially depleted in arsenopyrite. The excess As varies from 0.05 to 0.34 Wt % As, and is assumed to be due to As in solid solution in pyrite. Knowing the Wt % pyrite in each product (See Table 65) and the excess As, the inferred As content of pyrite is derived (See Table 67). With the exception of the Cu-scavenger and Cu-2nd cleaner concentrates, these vary from 0.17 to 0.79 Wt % As, and give a mean value of 0.31 Wt % As in pyrite. The range in excess As from 0.05 to 0.34 Wt %, together with the fact that this result is generated by material balance calculations on two independent datasets, both with moderate uncertainties, suggests that the uncertainty in the value of 0.31 Wt % As is high. This result should therefore be treated with considerable caution, until such time as it can be confirmed by microprobe analysis. Nonetheless, with the data available at present, the inference that Trout Lake pyrite contains a mean of 0.31 Wt % As, lends further support to the arguements indicating that 10-15 % of the Au in the ore occurs as 'invisible' Au in pyrite (less that in arsenopyrite).

TABLE 66

ADJUSTED As ASSAYS AND MODAL QUANTITIES OF ARSENOPYRITE AND RECOVERIES OF THESE IN THE 17 MILL PRODUCTS.

PRODUCT.	As		ARSENOPYRITE	
	ASSAY. (Wt %)	RECOVERY. (%)	MODAL AMOUNTS. (Vol %)	RECOVERY. (%)
1. Feed.	0.075	100.0	0.03	100.0
2. Cu-Scav Co.	0.283	7.3	0.59	37.7
3. Cu-Roug Co.	0.156	26.6	0.16	67.9
4. Cu-Cl.1 Fd.	0.179	32.6	0.17	76.6
5. Cu-Cl.1 Co.	0.090	9.6	0.09	23.7
6. Cu.Cl.2 Co.	0.038	3.6	0.06	15.0
7. Zn-Roug Fd.	0.084	106.5	0.04	114.0
8. Zn-Roug Co.	0.322	27.2	0.35	74.0
9. Zn-Cl.1 Co.	0.346	31.5	0.37	85.0
10. Zn-Cl.2 Co.	0.285	17.1	0.30	45.1
11. Cu-Roug Tl.	0.082	103.7	0.04	122.7
12. Cu-Scav Tl.	0.078	96.4	0.03	85.0
13. Cu-Cl.1 Tl.	0.303	23.0	0.28	52.9
14. Cu-Cl.2 Tl.	0.530	6.0	0.31	8.8
15. Zn-Cl.1 Tl.	0.413	10.1	0.48	29.0
16. Zn-Cl.2 Tl.	0.462	14.4	0.51	39.5
17. Zn-Roug Tl.	0.067	79.3	0.01	40.0

TABLE 67

DERIVATION OF INFERRED As CONTENT OF PYRITE IN THE 17 MILL PRODUCTS.

(Data in Weight %)

PRODUCT.	ARSENO-PYRITE MODAL AMOUNTS.	DERIVED As ASSAY.	ADJUSTED As ASSAY.	EXCESS As.	INFERRED As CONTENT OF PYRITE.
1. Feed.	0.060	0.026	0.075	0.05	0.33
2. Cu-Scav Co.	0.884	0.380	0.283	-0.10	----
3. Cu-Roug Co.	0.246	0.106	0.156	0.05	0.17
4. Cu-Cl.1 Fd.	0.268	0.115	0.179	0.06	0.20
5. Cu-Cl.1 Co.	0.146	0.063	0.090	0.03	0.19
6. Cu.Cl.2 Co.	0.101	0.043	0.038	-0.01	----
7. Zn-Roug Fd.	0.064	0.028	0.084	0.06	0.37
8. Zn-Roug Co.	0.530	0.228	0.322	0.09	0.38
9. Zn-Cl.1 Co.	0.562	0.242	0.346	0.10	0.30
10. Zn-Cl.2 Co.	0.453	0.195	0.285	0.09	0.71
11. Cu-Roug Tl.	0.071	0.030	0.082	0.05	0.31
12. Cu-Scav Tl.	0.053	0.023	0.078	0.06	0.38
13. Cu-Cl.1 Tl.	0.395	0.170	0.303	0.13	0.28
14. Cu-Cl.2 Tl.	0.439	0.189	0.530	0.34	0.79
15. Zn-Cl.1 Tl.	0.708	0.304	0.413	0.11	0.22
16. Zn-Cl.2 Tl.	0.774	0.333	0.462	0.13	0.21
17. Zn-Roug Tl.	0.028	0.012	0.067	0.06	0.35
Mean ($\bar{x}$)				0.08	0.31

LIBERATION OF THE AG-MINERALS FREIBERGITE, PYRARGYRITE, HESSITE AND ELECTRUM

The modal quantities, grain size distributions and liberations of freibergite, pyrargyrite, hessite and electrum in each of the 25 mill product samples were determined using the Minor Mineral Routine during image analysis. The modal quantities have been treated in MINERAL QUANTITIES ANALYSIS AND MINERAL BALANCES, whereas the liberations are briefly introduced for the first time here. Unfortunately, the counting statistics prevent any meaningful discussion of the grain size distributions of these minerals, and this data is omitted. Similarly, because of the small number of observations and hence extremely poor counting statistics for each of these minerals in each of the products, the liberation data in any of the products, and particularly the feed (the most significant product), are hopelessly inadequate for any reasonable conclusions to be made. However, by combining the data for each product, and tacitly assuming that the resulting errors in sample representivity are appreciably less than the counting statistics in the uncombined data, certain conclusions may be drawn. The modal quantities in raw surface area (in um^2) for both free and unliberated (i.e. Locked) freibergite, pyrargyrite, hessite and electrum are compiled in Table 68. The total surface area of free and unliberated grains of each mineral are also shown, as are the liberations calculated from the latter.

FREIBERGITE :

The total surface area of freibergite observed during image analysis of the 17 products is 9949 um^2, of which 893 um^2 occurs as free grains and 9056 um^2 occurs as unliberated grains; thus the calculated liberation of 9 %. However, it is apparent from Table 68, that 7450 um^2 of the total (i.e. 75 %) was recorded in the Cu-1st and 2nd cleaner tailings, in which high circulating loads have concentrated freibergite. As free grains of freibergite are expected to float, the high circulating loads of freibergite in the Cu-cleaner tailings preferentially consist principally of locked freibergite. Thus, observations from these two products strongly bias the resulting liberation. However, only the Cu-1st cleaner feed and the Cu-scavenger tailings contain a higher percent of free grains than unliberated grains, suggesting that the liberation of freibergite is most probably poor. Nonetheless, 96.5 % of the freibergite is calculated by MATBAL as being recovered in the Cu-concentrate. Because of the consistent association of freibergite and galena in the ores Healy & Petruk (1988) predicted that much of the unliberated freibergite would be locked with galena. By contrast however, as free galena is interpreted to be marginally floatable using Minerec, the recovery of these composite particles containing galena and freibergite is expected to be low.

PYRARGYRITE :

The total surface area of pyrargyrite observed is 414.9 um^2, of which 144.4 um^2 occurs as free grains and 270.5 um^2 occurs as unliberated grains; thus the calculated liberation of 35 %. It is apparent from Table 68, that the observations correspond to only 8 of the 17 products, and are both erratic and inherently uncertain. In addition, a mass of acanthite in the Zn-rougher concentrate (mis-identified by the Minor Mineral Routine) contributed to the

value of 96.8 um^2 locked pyrargyrite. This mass of acanthite reported by Healy & Petruk (1988) enveloped several grains of sphalerite and pyrite, and was associated with fine-grained (i.e. < 5 um) inclusions of silver-amalgam, both of which likely precipitated onto sphalerite from dissolved Ag and Hg in the pulp water.

Gasparrini (1984) reports that lime, which is used at Trout Lake, is deleterious to pyrargyrite flotation, and that pyrargyrite is sensitive to changes in flotation conditions. Thus, free pyrargyrite is not expected to be completely recovered in the Cu-concentrate, though from Table 68, it is apparent that at least a significant fraction is recovered. Unliberated pyrargyrite is probably locked with galena and/or pyrrhotite as predicted by Healy & Petruk (1988). Thus, the behaviour of this dominant fraction of unliberated pyrargyrite is a function of the combined effects of the uncertain floatability of pyrargyrite, and the degree of locking with galena and/or pyrrhotite. Nonetheless, with the exception of the 96.8 um^2 in the Zn-rougher concentrate (which is at least partly due to a mass of acanthite) unliberated pyrargyrite seems to report to the Cu-circuit in amounts greater than indicated by the material balance calculations (See Table 16).

HESSITE :

The total surface area of hessite observed is 956.4 um^2, of which 726.3 um^2 occurs as free grains and 230.1 um^2 occurs as unliberated grains; thus the calculated liberation of 76 %. Of the total surface area of hessite observed, 60 % was recorded in the Cu-1st and 2nd cleaner tailings, suggesting high circulating loads of hessite. As these are free grains, the floatability of hessite may be marginal, while the bias in sample representivity introduced by these observations, may have generated excessively elevated liberations. Certainly, a value of 76 % exceeds the expected liberation based on the fine-grained nature of hessite in the ore (Healy & Petruk 1988). Nonetheless, it is apparent that both free and unliberated hessite report principally to the Cu-circuit concentrates, with high circulating loads at the Cu-2nd cleaner tailings. It is noteworthy, that during image analysis of the mill products, virtually all observation of telluride minerals (i.e. hessite, volynskite, rucklidgeite and pilsenite) were recorded in the Cu-circuit concentrates.

ELECTRUM :

The total surface area of electrum observed is 2491 um^2, of which 1203 um^2 occurs as free grains and 1288 um^2 occurs as unliberated grains; thus the calculated liberation of 48 %. Of this 2491 um^2, 90 % of the observed grains were in the Cu-1st and 2nd cleaner concentrates and tailings, which not only biases the calculated liberation in terms of sample representivity, but also indicates that electrum is predominantly recovered in the Cu-concentrate, but with high circulating loads in the Cu-cleaning circuit. With the exception of one free grain (13.2 um^2) in the Zn-2nd cleaner tailings, all observations of electrum in the Zn-circuit are those of unliberated grains (See Table 68). This is in agreement with the interpretation that the principal form of electrum recovered in the Zn-concentrate is unliberated electrum occurring as inclusions in pyrite (i.e. type 1 electrum) that is in turn locked with

sphalerite. These are likely derived from the Massive Sphalerite and the Banded Pyrite + Sphalerite ore types.

Of the observations in the Cu-concentrate and the Cu-cleaner tailings, four large grains of electrum were observed i.e. 720 um^2, 518 um^2, 295 um^2 and 155 um^2, of which the first two are most probably type 2 electrum. The 720 um^2 grain observed in the Cu-concentrate appeared to be an elongate mass, typical of type 2 electrum (Healy & Petruk 1988), that had been milled, rolled and repeatedly bent back on itself (See Plate 1E and 1F. This grain was completely liberated and confirms that even relatively coarse-grained electrum (e.g. equivalent square diameter = 27 um), with a specific gravity of approximately 15, floats into the Cu-concentrate. Healy & Petruk (1988) concluded that the comparatively coarse-grained type 2 electrum undoubtedly represents the most volumetrically significant type, and even if unliberated and hence locked with chalcopyrite, should be largely recovered in the Cu-concentrate. The recovery of this free 720 um^2 grain in the Cu-concentrate, and the calculated (by MATBAL) recovery of 90 % of the electrum in the Cu-concentrate, tends to support those conclusions.

TABLE 68

RAW LIBERATION DATA FOR FREIBERGITE, PYRARGYRITE, HESSITE AND ELECTRUM IN THE 17 PRINCIPAL MILL PRODUCTS.

PRODUCT.	FREIBERGITE		PYRARGYRITE		HESSITE		ELECTRUM	
	Free Area (um^2)	Locked (um^2)	Free Area (um^2)	Locked (um^2)	Free Area (um^2)	Locked (um^2)	Free Area (um^2)	Locked (um^2)
1. Feed.	-----	-----	-----	-----	-----	-----	24.3	5.2
2. Cu-Scav Co.	94.5	188.5	3.9	5.0	23.1	147.4	20.0	22.0
3. Cu-Roug Co.	-----	150.0	-----	-----	-----	-----	-----	34.0
4. Cu-Cl.1 Fd.	7.05	4.6	6.6	-----	-----	-----	-----	-----
5. Cu-Cl.1 Co.	134.0	1036.	2.8	2.5	129.0	7.8	-----	59.7
6. Cu.Cl.2 Co.	344.3	406.6	12.0	159.0	-----	14.2	755.6	-----
7. Zn-Roug Fd.	-----	-----	-----	-----	-----	-----	-----	-----
8. Zn-Roug Co.	-----	97.7	21.2	96.8	-----	61.3	-----	69.8
9. Zn-Cl.1 Co.	-----	-----	-----	-----	-----	-----	-----	3.6
10. Zn-Cl.2 Co.	-----	8.3	-----	-----	-----	-----	-----	32.2
11. Cu-Roug Tl.	-----	-----	-----	-----	-----	-----	-----	-----
12. Cu-Scav Tl.	5.7	-----	-----	7.2	-----	-----	-----	6.6
13. Cu-Cl.1 Tl.	48.0	4462.	-----	-----	194.0	-----	295.0	722.0
14. Cu-Cl.2 Tl.	251.0	2689.	92.0	-----	372.0	-----	95.0	324.0
15. Zn-Cl.1 Tl.	-----	-----	-----	-----	-----	-----	-----	-----
16. Zn-Cl.2 Tl.	8.8	12.9	-----	-----	8.3	-----	13.2	-----
17. Zn-Roug Tl.	-----	-----	5.9	-----	-----	-----	-----	-----
TOTAL AREA (um^2)	893.	9056.	144.4	270.5	726.3	230.1	1203.	1288.
LIBERATION	9 %		35 %		76 %		48 %	

Chapter X

CONCLUSIONS

1. Barren waste particles comprise 55 % of the rod mill feed (1.25 inch passing). Of this fraction 60 % is greater than 10 mm in diameter, and hence 33 % of the bulk rod mill feed is in a size range potentially amenable to rejection by photometric sorting, or perhaps gravity separation of a specific size fraction of the rod mill discharge. Pre-concentration to reject part of this large waste fraction, if successful, would significantly lower processing costs, specifically grinding costs (See pages 14, 15, 111 and 112).

2. The proportion of the feed contributed by each of the 9 ore types and the barren wallrock, and their contribution to the assays of the valuable metals was calculated by material balances, and are as follows

Ore Type.	Weight	Cu	Zn	Au	Ag
Wallrock.	66.34	0.0	0.0	0.0	0.0
Cpy Strgs.	13.86	83.8	16.3	41.2	45.6
Msv Py.	11.37	1.6	21.4	16.8	8.9
M/B Cpy + Sph.	2.65	6.8	20.6	12.9	13.2
B Py + Sph.	2.22	0.5	15.6	9.4	16.5
Sh Cpy + Sph.	1.90	4.7	19.2	16.1	10.7
Remaining 4 ore types.	1.68	2.6	7.0	3.6	4.9

The large (i.e. 66 %) walllrock component is derived from (1) interbedded wallrock within the ore lenses; (2) dilution from wallrock within the mining width but not within the ore lenses; and/or (3) gangue occurring disseminated in the ore types, but not incorporated into the calculated composition of the ore types because of a bias in sampling towards gangue-poor, sulfide-rich samples. The second largest component (i.e. 13.86 %) of the feed is the Chalcopyrite Stringer ore type, which is the principal contributor of Cu (83.8 %), Au (41.2 %) and Ag (45.6 %). The third largest component (11.37 %) in the feed is the Massive Pyrite ore type, which contributed the largest proportion of Zn (i.e. 21.4 %) to the feed (See pages 11 to 14).

3. The grain size distributions of chalcopyrite, sphalerite and pyrite in the feed are 80 % - 26 um, 80 % - 36 um and 80 % - 55 um, respectively, while the particle size distribution of the feed (i.e. the grinding curve) is 80 % - 58 um. Grinding has resulted in the size reduction of chalcopyrite and sphalerite from 80 % - 175 um and - 208 um to 80 % - 26 um and - 36 um, respectively. However, the size reduction of pyrite from 80 % - 96 um to 80 % - 55 um is comparatively minor. The

shape of the grinding curve conforms to the size distribution of pyrite in the feed, which in turn parallels that of pyrite in the unbroken ore. During grinding the size distribution of sphalerite and chalcopyrite, pivot on the finer sizes, due principally to the dramatic decrease in the maximum grain size of these minerals, suggesting strong random breakage of coarse-grained chalcopyrite and sphalerite. In contrast, pyrite undergoes only a small decrease in maximum grain size, suggesting that pyrite breakage is dominated by preferential breakage (i.e. breakage along grain boundaries, fractures, cleavages and other rational crystallographic planes). Once having been ground by the process of preferential breakage to the size distribution of pyrite in the unbroken ore, the pyrite being a 'hard' mineral strongly resists further breakage, which is necessarily by the process of random breakage (See pages 126 to 131).

4. Material balance calculated composition of the studied feed in terms of the valuable metals, showed that the feed contained

2.16 % Cu	2.230 ppm Au	0.038 % Pb
2.57 % Zn	12.348 ppm Ag	0.006 % Cd

The feed contained average levels of Ag and Cu, low levels of Zn, Fe, S and Pb, and relatively high levels of Au relative to 'normal' feeds (See pages 16 to 25).

5. Material balance modelling of the raw chemical assays for the valuable metals Cu, Zn, Au, Ag, Cd and Pb through the circuit under simulated steady state conditions gives the following recoveries (%) :

	Cu-Conc.	Zn-Conc.	Tailings.
Cu	91.74	3.47	4.80
Zn	10.96	82.47	6.57
Au	59.22	15.25	25.54
Ag	48.49	21.74	29.77
Cd	11.26	73.96	14.79
Pb	13.09	45.52	41.40

(See pages 16 to 25).

6. Material balance modelling of the raw chemical assays for the metallurgically troublesome and environmentally hazardous elements Se, Sb, Sn, As and Hg through the circuit under simulated steady state conditions gives the following recoveries (%) :

	Cu-Conc.	Zn-Conc.	Tailings.
Se	26.44	7.34	66.23
Sb	18.44	26.17	55.39
Sn	65.47	23.20	11.33

	Cu-Conc.	Zn-Conc.	Tailings.
As	3.57	17.10	79.33
Hg	14.52	22.27	63.21

(See pages 16 to 25).

7. The distribution of the recoveries of each of the 18 chalcophilic elements analyzed indicates five principal groups of elements exhibiting relatively uniform behaviour, based on the flotation characteristics of the respective host mineral(s).

   a) Group 1 : Report principally to the tailings i.e. Mn, Ni, Co and Fe.

   b) Group 2 : Report largely to the tailings, with moderate recoveries in the Cu- and Zn-concentrates i.e. As, Sb, Hg, Se and S.

   c) Group 3 : Report principally to the Zn-concentrate i.e. Zn, Cd and Pb.

   d) Group 4 : Report principally to the Cu-concentrate, with moderate recoveries in the tailings and Zn-concentrate i.e. Au, Ag, Sn and In.

   e) Group 5 : Report principally to the Cu-concentrate i.e. Cu and Te.

   (See pages 16 to 25).

8. The metallurgically troublesome metal Bi was undetected (i.e. < 0.001 Wt %) in chemical assays of all 25 product samples (See page 16).

9. The mineralogical composition (in Vol %) of the studied feed was calculated by material balances to be

80.37 % Gangue (incl. silicates and carbonates)	
9.61 % Pyrite	
0.40 % Pyrrhotite	0.00011 % Electrum
4.60 % Chalcopyrite	0.00014 % Pyrargyrite
3.82 % Sphalerite	0.00011 % Gudmundite
0.03 % Arsenopyrite	0.00004 % Freibergite
0.024 % Galena	0.00004 % Hessite

(See pages 31 to 38).

10. The distribution of the recoveries of each of the 12 minerals whose modal quantities were determined, and which were material balance modelled through the circuit, indicate five principal groups of minerals each with relatively uniform flotation behaviour. These are

    a) Group 1 : Inert minerals which report principally to the tailings i.e. Gangue, pyrite and pyrrhotite.

b) Group 2 : Minerals that report principally to the Cu-concentrate i.e. chalcopyrite, electrum, freibergite and hessite.

c) Group 3 : Minerals with low recoveries in the Cu-concentrate, and with moderate to high recoveries in the Zn-concentrate and the tailings i.e. arsenopyrite, galena and gudmundite.

d) Group 4 : Minerals principally recovered in the Zn-concentrate i.e. sphalerite.

e) Group 5 : Minerals principally lost to the tailings, but with relatively high recoveries in the Cu-concentrate i.e. pyrargyrite.

(See pages 31 to 38).

11. The simplified classification of chalcopyrite in the feed, in terms of the principal liberation categories, is 60.6 % free, 29.8 % sub-free, 3.68 % middling and 3.27 % sub-middling grains, while 2.65 % of the chalcopyrite is unaccounted for on recombining the material balances of these categories. (See pages 68 to 81).

12. While 60.6 % of the chalcopyrite in the feed occurs as free grains (i.e. % liberation), 90.4 % of the chalcopyrite occurs as free plus sub-free grains. This large fraction of 'recoverable' categories of chalcopyrite implies excellent 'effective liberation', as these grains are expected to behave as free grains during flotation. While 99.3 % of the free grains, 97.3 % of the free plus sub-free grains are recovered in the Cu-concentrate, and is principally responsible for the overall recovery of 91.74 % of the Cu in the feed to the Cu-concentrate. The remainder of the recoverable categories of chalcopyrite is recovered in the Zn-concentrate (i.e. 0.59 %) and the tailings (i.e. 2.15 %). The minimal losses of 2.15 % of the recoverable categories of chalcopyrite, constitutes 1.94 % of the Cu in the feed, yet approximately 40 % of the Cu in the tailings. (See pages 68 to 81).

13. In addition to the 90.4 % recoverable categories of chalcopyrite in the feed, 3.68 % occurs in the middling category and 3.27 % in the sub-middling category. The sub-middling category (i.e. 'unrecoverable' category) of chalcopyrite reports principally to the Zn-concentrate (i.e. 47.6 %), largely as chalcopyrite-disease in sphalerite and diablastic and granular sphalerite-chalcopyrite particles. However, this only accounts for 1.6 % of the chalcopyrite in the feed, yet constitutes approximately 45 % of the chalcopyrite in the Zn-concentrate. Similarly, 37.4 % of the sub-middling chalcopyrite reports to the tailings, and accounts for 1.2 % of the chalcopyrite in the feed, yet approximately 25 % of the chalcopyrite in the tailings. These combined losses of approximately 2.8 % of the Cu in the form of sub-middling chalcopyrite are essentially unrecoverable through regrinding. (See pages 68 to 81).

14. The 3.68 % of the chalcopyrite in the feed that occurs in the middling category is estimated (i.e. poor material balance for this category) to report to the Cu-concentrate (i.e. 50 %), the Zn-concentrate (i.e.

20 %) and the tailings (i.e. 30 %). The middling chalcopyrite lost to the tailings constitutes approximately 1.1 % of the chalcopyrite in the feed, yet constitutes approximately 35 % of the chalcopyrite in the tailings. Similarly, the middling chalcopyrite lost to the Zn-concentrate consitutes 0.7 % of the chalcopyrite in the feed, yet approximately 30 % of the chalcopyrite in the Zn-concentrate. These combined losses of approximately 1.8 % of the Cu in the feed in the form of middling chalcopyrite are amenable to regrinding, and hence improved liberation and recovery. (See pages 68 to 81).

15. The Cu-1st cleaner tailings (stream flow rate = 5.67) contains 51.9 % of the middling chalcopyrite, whereas the Cu-scavenger concentrate (stream flow rate = 1.93) contains 22.1 % of the middling chalcopyrite. In both streams approximately 30 % of the unliberated chalcopyrite is locked with sphalerite. By combining these streams (both are presently returned to the feed) into a single stream approximately 74 % of the middling chalcopyrite (the principal category amenable to regrinding) may be presented to the regrind with only a slight change in the circuit configuration. However, because only 3.68 % of the Cu in the feed occurs as middling grains of chalcopyrite, and as only 50 % of this is lost to the Zn-concentrate and tailings (combined), the improvement in Cu recovery due to enhanced liberation from middling chalcopyrite is likely to be small i.e. approximately 1 %. (See pages 68 to 81).

16. The principal types of chalcopyrite lost to the tailings are chalcopyrite-pyrite, chalcopyrite-gangue and chalcopyrite-pyrrhotite particles. The principal types of chalcopyrite lost to the Zn-concentrate are (1) chalcopyrite-disease in sphalerite; (2) diablastic and granular sphalerite-chalcopyrite particles; (3) middling chalcopyrite-pyrite particles that failed to float in the Cu-circuit; and (4) coarse-grained (i.e. 'over-sized') sub-free (particularly in the 90-99.9 class) grains of chalcopyrite. With the exception of (1) these types of chalcopyrite-bearing particles, and particularly over-sized sub-free chalcopyrite and diablastic sphalerite-chalcopyrite, are concentrated in the Cu-scavenger concentrate. Thus, regrinding the proposed combined stream will bring much of the over-sized chalcopyrite into the size range for flotation (i.e. < 100 um), and improve the Cu-Zn separation by preferential liberation from chalcopyrite-sphalerite particles. Thus, the regrinding of over-sized sub-free chalcopyrite (which is clearly amenable to regrinding) will lower the overall losses of sub-free chalcopyrite, which presently comprise 10-20 % of the chalcopyrite lost to the Zn-concentrate and approximately 40 % of the chalcopyrite lost to the tailings. (See pages 68 to 81).

17. The grain size of free chalcopyrite in the feed is 80 % - 21 um, whereas that of unliberated chalcopyrite is 80 % - 28 um. While the grain sizes of these in the Cu-concentrate are 80 % - 24 um and 80 % - 28 um, respectively, those in the Cu-scavenger concentrate are by contrast 80 % - 67 um and 80 % - 77 um, respectively. Furthermore, the particle size distribution of the Cu-scavenger concentrate is 80 % - 70 um, making it the coarsest-grained of the 17 products. As 76 % of the chalcopyrite in the Cu-scavenger concentrate is

unliberated, of which 55 % is locked with sphalerite, this product is highly suitable for regrinding. This should achieve a decrease in the fraction of over-sized chalcopyrite, and an increase in chalcopyrite-sphalerite separation. (See pages 126 to 144).

18. The simplified classification of sphalerite in the feed in terms of the principal liberation categories is 50.5 % free, 41.0 % sub-free, 4.56 % middling and 1.86 % sub-middling grains, while 2.08 % of the sphalerite is unaccounted for on recombining the material balances of these categories. (See pages 81 to 98).

19. While 50.5 % of the sphalerite in the feed occurs as free grains (i.e. % liberation), 91.5 % of the sphalerite occurs as free plus sub-free grains. This large fraction of 'recoverable' categories of sphalerite implies an excellent 'effective liberation', as these grains are expected to behave as free grains during flotation. While 90.7 % and 9.29 % of the free grains report to the Zn-concentrate and Cu-concentrates, respectively, 86.7 % and 10.3 % of the free plus sub-free grains report to these same concentrates, respectively. The tailings recover 0.01 % of the free grains of sphalerite, but recover 3.01 % of the free plus sub-free grains of sphalerite. In all, 96.7 % of the recoverable categories of sphalerite entering the Zn-circuit are recovered in the Zn-concentrate, indicating that Cu-activation by $CuSO_4$ and flotation by Minerec in the Zn-circuit is operating relatively efficiently. Because the recoverable categories of sphalerite constitute 91.5 % of the Zn in the feed, the minimal loss of 3.01 % of these categories to the tailings results in a corresponding loss of 2.75 % of the Zn in the feed, but which constitutes approximately 40 % of the Zn lost to the tailings. (See pages 81 to 98).

20. In addition to 91.5 % recoverable categories of sphalerite in the feed, 4.56 % occurs as middling grains. The middling category of sphalerite reports principally to the Zn-concentrate (i.e. 65 %), and constitutes 2.97 % of the Zn in the feed. The Cu-concentrate recovers 19.5 % of the middling sphalerite, and constitutes 0.9 % of the Zn in the feed, and approximately 8 % of the Zn in the Cu-concentrate. The tailings recovers 15.4 % of the middling sphalerite, which corresponds to 0.7 % of the Zn in the feed, and constitutes approximately 30 % of the Zn in the tailings. As only 4.56 % of the Zn in the feed occurs as middling grains of sphalerite, of which only 45 % is lost to the Cu-concentrate and tailings combined, and which is the principal category of particle amenable to regrinding, enhanced liberation and recovery of sphalerite by regrinding is expected to be relatively low i.e. 1 or 2 %. (See pages 81 to 98).

21. Of the 1.86 % of the sphalerite in the feed that occurs in the sub-middling category, 0.72 % (i.e. approximately 39 %) is recovered in the Zn-concentrate, 0.42 % (i.e. approximately 22 %) is recovered in the Cu-concentrate and 0.72 % (i.e. approximately 39 %) is recovered in the tailings. The losses of sub-middling sphalerite to the Cu-concentrate constitute < 5 % of the sphalerite in the Cu-concentrate, whereas those lost to the tailings constitute approximately 30 % of the sphalerite in the tailings. These combined losses of approximately 1.1 % of the Zn in

the feed in the form of 60 % of the sub-middling sphalerite, are essentially unrecoverable through regrinding. (See pages 81 to 98).

22. While 11 % of the Zn in the feed is recovered in the Cu-concentrate, the classification of this in terms of the liberation categories of the constituent sphalerite is 4.70 % free, 4.56 % sub-free, 0.89 % middling and 0.42 % sub-middling grains, while 0.43 % remained unaccounted for on recombining the material balances of these categories. Thus, of the sphalerite in the Cu-concentrate, 43 %, 42 %, 8 % and 4 % occur as free, sub-free, middling and sub-middling grains of sphalerite, respectively. Material balancing of the recoverable categories of sphalerite combined gave 10.3 % recoveries in the Cu-concentrate, which corresponds to 9.4 % of the Zn in the feed, and approximately 86 % of the Zn in the Cu-concentrate. Thus, 86 % of the Zn/sphalerite lost to the Cu-concentrate is recoverable (in the Zn-concentrate), confirmimg that these Zn losses are not due to poor liberation, but instead due to poor rejection of sphalerite in the Cu-circuit. (See pages 81 to 98).

23. In the Cu-scavenger and Cu-rougher stages, the flotation efficiencies for unliberated sphalerite exceed those of free sphalerite. However, through Cu-cleaning the cumulative flotation efficiencies for free sphalerite (i.e. 39.1) is greater than that of unliberated sphalerite (i.e. 34.4), suggesting preferential recovery of free sphalerite. Thus, Cu-cleaning preferentially rejects unliberated sphalerite locked with minerals other than chalcopyrite, and concentrates free sphalerite and unliberated sphalerite locked with chalcopyrite. Thus, 36 %, 80 %, 89 % and 88 % of the unliberated sphalerite in the feed, Cu-rougher concentrate, the Cu-1st cleaner concentrate and the Cu-2nd cleaner concentrate, respectively is locked with chalcopyrite. However, 76 % of the unliberated sphalerite in the Cu-concentrate occurs as sub-free grains, which irrespective of being locked with chalcopyrite, are expected to behave essentially as free grains of sphalerite and report to the Zn-concentrate. By corollary, a maximum of only 13 % of the sphalerite in the Cu-concentrate can be explained on the basis of poor liberation, and occurring in particles locked with chalcopyrite. Thus, subordinate (i.e. comprising < 30 % of the host particle) chalcopyrite locked with sphalerite (i.e. sub-free sphalerite) exerts a strong influence on the latters flotation behaviour, and which is disproportionate with respect to its volume fraction of the particle. It is therefore concluded that sphalerite is marginally floatable, and that subordinate locked chalcopyrite may exert sufficient influence on flotation, so as to carry the sphalerite grain over the threshold for flotation. (See pages 81 to 98).

24. The cumulative flotation efficiencies through Cu-cleaning of both free (i.e. 39.1 %) and unliberated (i.e. 34.4) sphalerite greatly exceed that of pyrite (i.e. 19.7). In addition, while 14.4 % of the sphalerite (or 11 % of the Zn; the discrepancy being due to independent material balances), only 0.54 % of the gangue and 6.36 % of the pyrite in the feed are recovered in the Cu-concentrate. The proportionally low pyrite and gangue content indicates that the Cu-concentrate is essentially 'clean', despite the high sphalerite content. The high rejection of pyrite and gangue from the Cu-circuit therefore indicates

that the apparent lack of self-draining or 'windowing' of the air bubbles in the froth, due possibly to the frothing properties of the Minerec collector, is not the cause of the Zn losses to the Cu-concentrate. (See pages 180 to 184).

25. The trace amounts of the potential activator mineral covellite (< 0.01 Wt %) in the float feed are derived from the alteration of Cu- and Cu-Fe minerals, which formed by equilibration of entrained Cu-matte in slag. Thus, covellite is introduced into the feed from extraneous slag used as mine backfill, and which comprises part of the mine dilution. It is also possibly introduced into the feed via the secondary cone crusher, which is used to crush slag for various construction and backfill purposes. While the introduction of the slag and associated covellite into the feed is strongly discouraged, the low modal abundances of covellite (< 0.01 Wt %), and the lack of response of sphalerite flotation to cyanidation of the pulp water (i.e. not suppressed), militate against covellite activating sphalerite, and causing the high Zn losses to the Cu-concentrate. (See pages 172 to 179).

26. The modal abundances of the reactive mineral marcasite in the feed is 0.17 Wt %. All observed grains seemed fresh and unaltered, suggesting that the fast turnover of essentially dry ore (i.e. 1 to 3 days) minimizes oxidation of the marcasite, and its potential role in the activation of sphalerite. Ferrous ions in solution in the presence of dixanthogen and at a pH range of 9.5 to 11, can cause flotation of sphalerite without Cu-activation. However, it is not known whether oxidized Minerec (if present) exhibits similar behaviour, while the low modal abundances of, and the fresh, unaltered character of, the marcasite militate against it releasing sufficient ferrous ions into solution to have a significant effect on sphalerite flotation. (See page 171).

27. The grain size distribution of free sphalerite in the feed is 80 % - 22 um (with 7.5 % in the - 5 um or slime fraction), whereas the grain size distribution of free sphalerite in the Cu-concentrate is 80 % - 17 um (with 13.5 % in the slime fraction). Thus, while free sphalerite in the Cu-concentrate is slightly finer-grained than that of the feed, only 13.5 % occurs in the slime fraction, and potentially recovered by mechanical entrainment of sphalerite slimes in the froths of the Cu-circuit. The grain size distribution of unliberated sphalerite in the feed is 80 % - 48 um (with approximately 1 % slime fraction), whereas the grain size distribution of unliberated sphalerite in the Cu-concentrate is 80 % - 20 um (with 9.5 % slime fraction). Thus, while unliberated sphalerite in the Cu-concentrate is also finer-grained than that in the feed, it contains a relatively minor (i.e. 9.5 %) slime fraction. Consequently, it is concluded that the entrainment of sphalerite slimes contributes negligibly to the Zn losses in the Cu-concentrate, but that the principal causative mechanism of these Zn losses preferentially recovers slightly finer fractions of both free and unliberated sphalerite. (See pages 135 to 147).

28. The mean composition of 47 microprobe analyses of sphalerite in the feed is 59.02 % Zn, 7.02 % Fe, 33.14 S, 0.27 % Cd and 0.05 % Cu (total = 99.68). This compares with that of 52 microprobe analyses of sphalerite in the Cu-concentrate, which is 59.28 % Zn, 6.91 % Fe, 33.07 % S, 0.23 % Cd and 0.49 % Cu (total = 99.98). While this suggests that Trout Lake sphalerite contains a natural range of Cu in solid solution, and that the Cu-concentrate preferentially recovers the Cu-rich species, the compositions can also be explained simply on the basis of analytical contamination. A second set of 26 microprobe analyses of sphalerite in a section of the Cu-concentrate (specially prepared to minimize analytical contamination) gave a mean composition of 59.60 % Zn, 6.77 % Fe, 33.23 S, 0.20 % Cd and 0.35 % Cu (total = 100.20). On the basis of this and other experimental work, and theoretical considerations of analytical contamination from Cu(Kα) in adjacent chalcopyrite grains, and the known solid solubility of Cu in sphalerites, it is tentatively concluded that the apparent enrichment of Cu in sphalerite in the Cu-concentrate is spurious, and that the Cu contents of sphalerite are considerably less than measured and play no discernible role in sphalerite flotation. However, this work failed to unequivocally characterise the composition, specifically the Cu content, of the sphalerite in the Cu-concentrate. (See pages 167 to 171).

29. Poor sphalerite liberation, entrainment of sphalerite slimes, activation, poor self-draining of the air bubbles in the froth, and the influence of Cu in solid solution in sphalerite are all either rejected or disfavoured as the mechanism of the high Zn losses to the Cu-concentrate. It is therefore tentatively concluded that while pyrite is very inert to, and is readily rejected by Minerec (which is specifically used for the selective flotation of copper sulfides from pyrite), Minerec lacks the selectivity for efficient chalcopyrite-sphalerite separation. Hence, sphalerite is marginally floatable in the Cu-circuit prior to activation with $CuSO_4$. The finer-grain size of sphalerite in the Cu-concentrate relative to the feed, and the conspicuous influence of subordinate (< 30 % of the particle) locked chalcopyrite on the sub-free sphalerite, are compatible with such marginal flotation characteristics.

30. The table below summarises certain key values discussed under this conclusion, which relates to the merits of the proposed versus the present regrind stream.

The Cu-scavenger concentrate recovers 7.89 % of the recoverable (i.e. free plus sub-free) categories and approximately 15 % of the middling category of sphalerite, corresponding to 7.22 % and 0.7 %, respectively of the Zn in the feed. The recoverable categories of sphalerite comprise 87 % of the Zn in the Cu-scavenger concentrate, of which only 15 % occurs as free grains. Furthermore, 94 % of the unliberated sphalerite in the Cu-scavenger concentrate is locked with chalcopyrite.

The Cu-1st cleaner tailings recovers 14.7 % of the recoverable categories and approximately 25 % of the middling category of

sphalerite, corresponding to 13.5 % and 1.4 %, respectively of the Zn in the feed. The recoverable categories comprise 85 % of the Zn in the Cu-1st cleaner tailings, of which 40 % occurs as free grains. Furthermore, 88 % of the unliberated sphalerite in the Cu-1st cleaner tailings is locked with chalcopyrite.

Parameter.	Cu-Scav Co.	Cu-1st Cl Tl.	Combined.
Stream flow rate.	7.89	5.67	7.60
Zn assay (Wt %).	11.1	7.18	9.46
Zn recovery (%).	8.32	15.8	24.2
Recovery of 'recoverable' Sphal.	7.89	14.7	22.6
Fraction of Zn in the feed occurring as 'recoverable' Sphal.	7.22	13.5	20.7
Fraction of Zn occurring as 'recoverable' Sphal.	87	85	86
Fraction of 'recoverable' Sphal as free Sphal.	15	40	32
Recovery of middling Sphal.	15	25	40
Fraction of Zn in feed occurring as middling Sphal.	0.7	1.4	2.1
Fraction of unliberated Sphal locked with Cpy.	94	88	90

A combined stream of the Cu-scavenger concentrate and the Cu-1st cleaner tailings (with a stream flow rate = 7.60) therefore contains 24.2 % of the Zn in the feed, of which 27 % occurs as free, 59 % as sub-free, 9 % as middling and 4 % as sub-middling grains of sphalerite. This compares with the presently reground stream (i.e. 60 % of the Cu-rougher tailings; flow rate = 57) which contains 58.4 % of the Zn in the feed, of which approximately 50 % occurs as free, 42 % sub-free, 4 % as middling and 4 % as sub-middling grains of sphalerite. The present reground split stream of the Cu-rougher tailings therefore contains a higher fraction of the sphalerite in the feed. (See pages 87 to 98).

Whereas the proposed regrind stream should contain 45 % of the middling sphalerite in the feed, the split Cu-rougher tailings regrind stream contains 57 % of the middling sphalerite in the feed. However, the latter occurs diluted in a stream enriched in the 'hard' minerals pyrite (i.e. recovery of 101.2 % versus 7.58) and gangue (i.e. recovery of 99.99 % versus 0.53), and whose stream flow rate is an order of magnitude greater. As the particle size distribution of a ground product is largely controlled by the grain size distribution of the constituent pyrite and gangue (e.g. quartz), regrinding the proposed combined stream, which contains low recoveries of pyrite and gangue, and whose pyrite and gangue contents are predominatly unliberated, should produce a significant decrease in grain size and/or a reduction in the grinding costs. Thus, for the equivalent size reduction the

potential for enhanced liberation of chalcopyrite and sphalerite is significant. Because of the overwhelming predominance of recoverable categories of chalcopyrite and sphalerite in the feed (i.e. 90.4 % and 91.5 %, respectively), and the accordingly minor middling categories (i.e. 3.68 % and 4.56 %, respectively) it might be expected that regrinding will generate relatively small improvement (i.e. 1 or 2 %) in the liberation and recoveries of Cu and Zn. However, regrinding should also address the 'over-sized' free and sub-free chalcopyrite, as well as the subordinate chalcopyrite locked with sub-free sphalerite, which seems to have a disproportionate effect on the latters flotation behaviour. Thus, regrinding should improve both Cu and Zn recoveries over that nominally expected from the middling category, and on the basis of the simplified four-fold classification of chalcopyrite and sphalerite liberation. (See pages 68 to 98 and 163 to 166).

31. Diablastic-textured (i.e. fine-grained carious to myrmekitic interpenetrative intergrowths) particles of chalcopyrite-sphalerite comprise approximately 20-25 % of the unliberated sphalerite in the Cu-concentrate. In addition, conspicuous diablastic sphalerite-chalcopyrite and sphalerite-pyrrhotite particles were observed in the Zn-concentrate. This texture is derived principally from the Sheared Chalcopyrite + Sphalerite ores, which dominate the No. 2 lens of the north zone, but which were calculated as comprising only 2 % of the feed. The persistance of a significant but unknown fraction of the diablastic chalcopyrite-sphalerite through grinding implies that liberation from this texture is probably low, and that the two principal valuable metals Cu and Zn are partly inseparable due to this poor liberation. This liberation-dependent mechanism of poor Cu and Zn separation is expected to be pronounced for feeds rich in the Sheared Chalcopyrite + Sphalerite ores, and particularly where the No. 2 lens of the north zone comprises all or part of the feed. As lens No. 2 of the north zone is the largest and richest of the proven ore lenses, as it is still open at depth, and as the diablastic texture is projected to be at least discontinuously developed to depth, significant losses of Zn (in the form of unliberated sphalerite locked with chalcopyrite) to the Cu-concentrate are predicted. (See pages 163 to 166).

32. Comparison of the particle size distributions of the Cu-rougher tailings to that of the combined Cu-scavenger concentrate and tailings (i.e. regrind stream reconstituted) gave a size reduction from 80 % - 58 um to 80 % - 50 um. This constitutes less than a mesh size reduction in particle sizes, or at 43 um (i.e. 325 mesh) a 3 % decrease in particle sizes. Such a minor size reduction is not expected to yield significant improvement in liberation of unliberated chalcopyrite or sphalerite, though it may 'clean' or 'freshen-up' the mineral grain surfaces. The Cu-rougher tailings contains 84.7 % gangue and 10.25 % pyrite, corresponding to 99.9 % and 101.2 % of the gangue and pyrite, respectively in the feed. Thus, much of the energy expended is consumed grinding 'hard' waste minerals destined for the tailings, and which are already principally free of valuable minerals. (See pages 120 to 121).

33. Chalcopyrite-disease, which is relatively ubiquitous in the ores, persists in the ground product, occurring in approximately 5 % of the sphalerite grains in the feed. While the bulk of the persistent chalcopyrite-disease is in the form of encapsulated chalcopyrite blebs and rods in sphalerite, a minor component is in the form of chalcopyrite blebs and rods lying along the grain edges of the sphalerite, and hence exposed to the flotation medium. Both varieties 'seem' to be preferentially concentrated, perhaps by a factor of 3, in the Cu-concentrate relative to that in the feed. Nonetheless, as 87.8 % of the 90-99.9 (i.e. essentially free) class of unliberated sphalerite (which includes most of the persistent chalcopyrite-disease) is recovered in the Zn-concentrate, and as chalcopyrite-disease is approximately half as frequent in the Zn-concentrate as in the Cu-concentrate, the bulk of the mass of sphalerite containing chalcopyrite-disease is recovered in the Zn-concentrate. Thus, chalcopyrite-disease is not considered as a principal contributor to the losses of recoverable categories of sphalerite to the Cu-concentrate. Because approximately 20 % of the unliberated sphalerite in the Cu-concentrate contains chalcopyrite-disease, it merely constitutes another symptom of the 'ailment' i.e. that sphalerite is marginally floatable and that accompanying locked chalcopyrite provides the requisite flotation to carry the sphalerite grain above the threshold for flotation. (See page 164).

34. The liberation of galena in the feed is extremely low, with only 7.69 % occurring as free grains, and 92.3 % occurring as unliberated grains. The small component of free galena 'seems' to be largely, but not wholely, recovered in the Cu-concentrate. Furthermore, sub-free grains of galena, which are expected to behave as free grains during flotation, report principally to the Zn-concentrate (i.e. 49.5 %) and the tailings (i.e. 38.8 %), such that Minerec seems to achieve relatively poor flotation of nominally recoverable grains of galena. The unliberated galena reports principally to the Zn-concentrate (i.e. 41.8 %) principally as sphalerite-galena particles, and the tailings (i.e. 47.0 %) principally as pyrite-galena particles. Thus, Pb and the minor component of the Ag assay in solid solution in galena are recovered principally in accordance with the flotation characteristics of the host minerals of unliberated galena. (See pages 99 to 108).

35. Mercurian electrum, the principal Au-carrier, contains approximately 85 to 90 % of the Au in the ore. While the liberation of electrum is 'very' approximately 50 %, the recovery of electrum in the Cu-concentrate is 90 %. This probably stems from the predominance of type 2 as the most volumetrically siginficant type of electrum, which if unliberated and hence locked with chalcopyrite, is expected to report to the Cu-concentrate. Though no electrum was observed in the tailings, unliberated type 1 electrum was observed in several products. MATBAL calculated losses of 7 % electrum to the tailings, probably as un-observed type 1 electrum locked in pyrite. Several large grains of free type 2 electrum in the Cu-circuit concentrates, and confirms that electrum (mercurian) floats using Minerec collector, though with high circulating loads of electrum during Cu-cleaning. (See pages 190 to 195 and 201 to 202).

36. It is inferred that approximately 10 to 15 % of the Au in the ore is carried as 'invisible' Au, most probably in solid solution, in pyrite and arsenopyrite. As 89 % of the pyrite and 40 % of the arsenopyrite is lost to the tailings, approximately 13 % of the Au in the ore is lost to the tailings as 'invisible' Au in these minerals. This is in general agreement with cyanidation tests of the tailings, which indicate that 50 to 60 % of the Au in the tailings is 'invisible' or 'refractory', and therefore is not amenable to recovery without destruction of these host minerals e.g. by roasting, double oxidation, chlorination, etc. In the absence of direct experimental measurement, the fractions of the 'invisible' Au hosted in pyrite and arsenopyrite is unknown, such that their mean Au contents potentially vary from 0-2 and 0-1600 ppm Au, respectively. It may also be concluded that 'invisible' Au in arsenopyrite comprises a significant fraction of the Au reporting to the Zn-concentrate, the principal repository of arsenopyrite. (See pages 190 to 195).

37. An excess of Ag over that which can be accounted for by the material balancing of the modal quantities of the known Ag-bearing minerals electrum, freibergite, pyrargyrite, hessite and the ultra-trace Ag-minerals volynskite, freieslebenite, naumannite and acanthite was observed in all products. In the absence of direct measurement, it is inferred from material balance calculations that chalcopyrite, sphalerite, pyrite and galena contain 45, 55, 11 and 750 ppm Ag, respectively, which corresponds to 23 %, 19 %, 18 % and 2.7 % of the Ag in the feed. Electrum, pyrargyrite, freibergite and hessite account for 18 %, 13 %, 6 % and 5 %, respectively of the Ag in the feed. As sphalerite contains 19 % of the Ag in the feed, resolution of the 'sought after' improvement in Zn recovery to the Zn-concentrate will result in routine losses of approximately 20 % of the Ag to the Zn-concentrate. Thus, Zn and Ag are partly inseparable during flotation. As 14 % of the Ag in the ore occurs in pyrite, 88.8 % of which is recovered in the tailings, approximately 12 % of the Ag in the ore is expected to routinely report to the tailings in a form that is economically unrecoverable by flotation. Furthermore, this 'refractory' Ag in solid solution in pyrite accounts for 40 to 50 % of the Ag in the tailings. The loss of 58 % of the pyrargyrite in the feed to the tailings, results in losses of 7.5 % of the Ag in the ore, and corresponds to 25 % of the Ag in the tailings. The high recoveries (i.e. in the 90's except for hessite) of chalcopyrite, electrum, freibergite and hessite in the Cu-concentrate account for 86 % of the Ag in the Cu-concentrate, and corresponds to 40 to 45 % of the Ag in the ore. (See pages 185 to 190).

38. It is inferred from material balance calculations on the As assays, and arsenopyrite and pyrite modal quantities that the excess As (above that which can be explained by arsenopyrite) occurs in solid solution in pyrite. The mean As content of Trout Lake pyrite is inferred to be 0.31 Wt % As. This further supports the inference of 10 to 15 % 'invisible' Au in pyrite (less that in arsenopyrite), in view of the common association of 'invisible' Au with arsenian pyrite. (See pages 195 to 199).

Chapter XI

RECOMMENDATIONS

1. Investigate possible pre-concentration techniques e.g. photometric ore sorting or gravity separation, to remove the large fraction of barren waste particles in the rod mill feed, or perhaps a specific size fraction of the rod mill discharge. The latter however, would be less attractive as some 'wasteful' grinding would have been already performed by that stage.

2. Discontinue backfilling with slag where stopes are likely to be reactivated, and minimize all possible opportunity of presently backfilled slag from being introduced into the feed through mine dilution.

3. Investigate the effect of crushing slag on the crushing surfaces of the secondary cone crusher (i.e. smearing of Cu- and Cu-Fe-sulfides), and any possible introduction of slag into related ore bins. Discontinue the practise if contamination is apparent.

4. Investigate regrinding (to perhaps 80 % - 25 or 20 um) a combined stream of the Cu-scavenger concentrate and the Cu-1st cleaner tailings, either in conjunction with the present regrind of the split Cu-rougher tailings, or solely, but with the Cu-scavenger 'pulling extremely hard'. The latter is especially important where Sheared Chalcopyrite + Sphalerite ore from lens No. 2 of the north zone is conspicuous in the feed.

5. Conduct bench tests on the float feed to determine the optimum reagent conditions, and particularly the best collector or combination of collectors (e.g. long chain Xanthates plus Minerec) for optimum chalcopyrite and sphalerite recovery, and suppression of the sphalerite which is presently reporting to the Cu-circuit, due to poor selective flotation.

6. Where significant losses to the tailings of Zn, Ag and Pb are experienced during processing of ore from the No. 5 lens of the south zone (e.g. stope No. 2451), sample the feed, the tailings, the Zn-rougher concentrate, the Zn-1st cleaner tailings and the Zn-concentrate. Forward them to the author for petrographic examination. If as expected, the losses are due to dendritic sphalerite, galena and freibergite in pyrite, the pull on the Zn-rougher should be increased and the Zn-1st cleaner tailings should be reground, either in a separate mill, or by returning it to the proposed regrind stream; thus recover the liberated galena and freibergite in the Cu-circuit, while

the sphalerite will re-enter the Zn-circuit, provided sphalerite is desorbed of Cu by further conditioning.

7. Conduct proton microprobe analysis (PIXE) of Ag in chalcopyrite, sphalerite and pyrite, to verify the inferences made in this report pertaining to the principal Ag-carriers.

8. Conduct secondary ion mass spectrometric analysis (SIMS) of Au in pyrite and arsenopyrite, to confirm the character of the 'invisible' Au, and to determine the mean Au content of both minerals.

9. Conduct electron microprobe analysis (high precision using WDA) of As in pyrite, to verify its inferred arsenian character. This work can be performed by the author at the University of Manitoba, on receipt of the new Cameca SX-50 microprobe.

10. Routinely monitor the mineralogical nature of the sphalerite reporting to the Cu-circuit concentrates (particularly the Cu-2nd cleaner concentrate) by petrographic examination using the Reichart Metallograph microscope housed in the HBM&S assay laboratory, Flin Flon.

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Appendix A

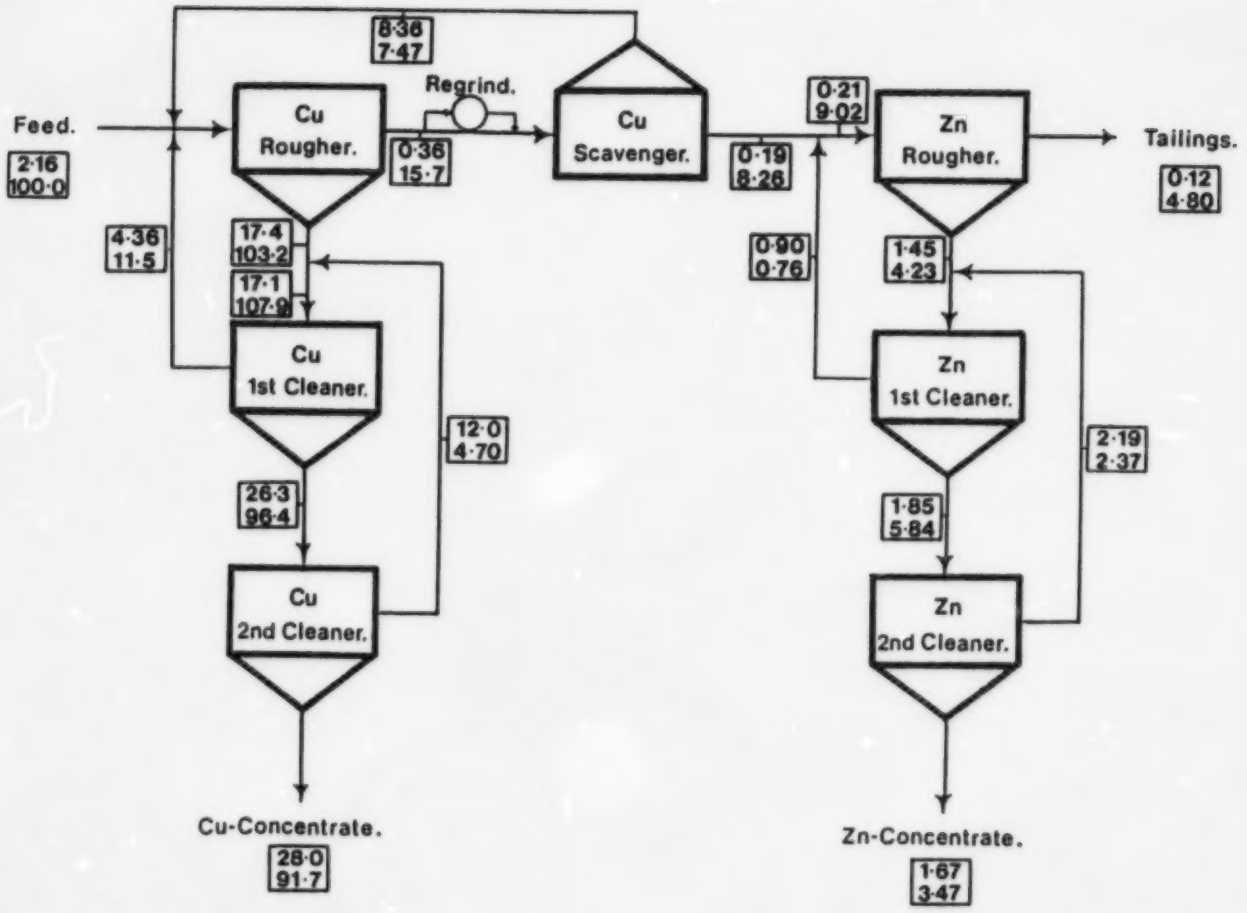

FIGURE 1.
THE TROUT LAKE CONCENTRATOR CIRCUIT DIAGRAM PORTRAYING THE ADJUSTED ASSAYS AND RECOVERIES FOR Cu IN EACH PRODUCT. VALUES IN THE BOXES ARE (TOP) MATERIAL BALANCE ADJUSTED ASSAYS (WT %), AND (BOTTOM) THE CORRESPONDING RECOVERIES (%).

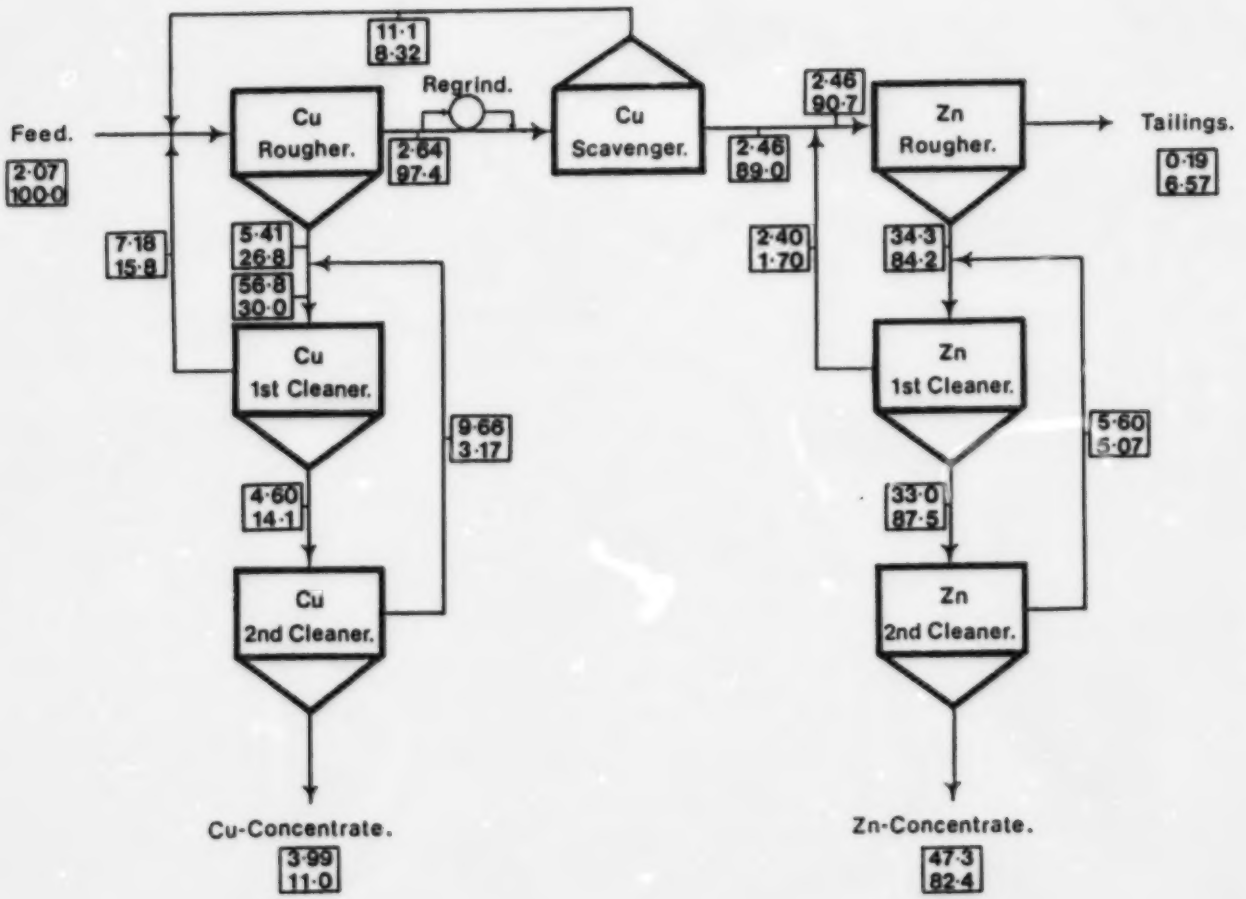

FIGURE 2.
THE TROUT LAKE CONCENTRATOR CIRCUIT DIAGRAM PORTRAYING THE ADJUSTED ASSAYS AND RECOVERIES FOR Zn IN EACH PRODUCT. VALUES IN THE BOXES ARE (TOP) MATERIAL BALANCE ADJUSTED ASSAYS (WT %), AND (BOTTOM) THE CORRESPONDING RECOVERIES (%).

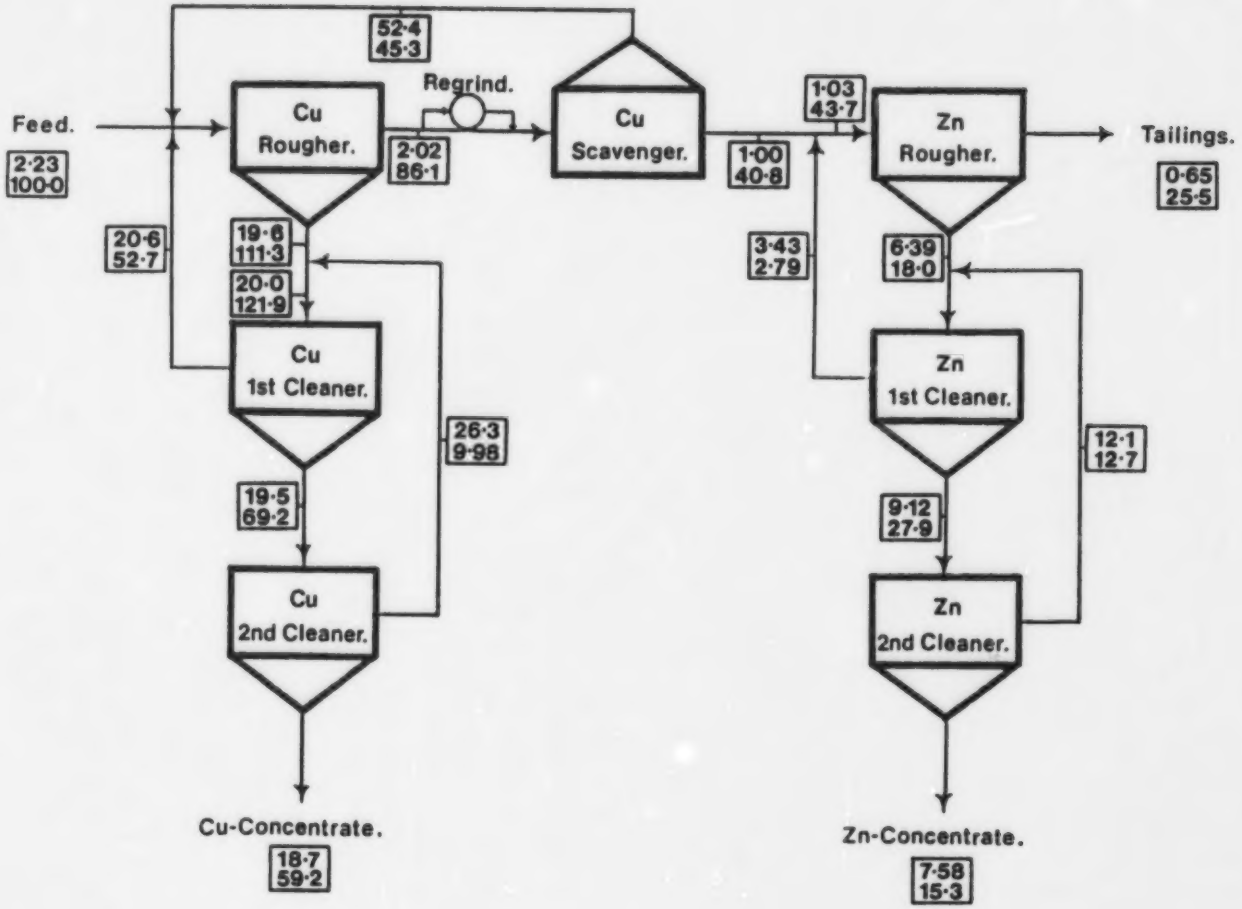

FIGURE 3.
THE TROUT LAKE CONCENTRATOR CIRCUIT DIAGRAM PORTRAYING THE ADJUSTED ASSAYS AND RECOVERIES FOR Au IN EACH PRODUCT. VALUES IN THE BOXES ARE (TOP) MATERIAL BALANCE ADJUSTED ASSAYS (OZ/TON), AND (BOTTOM) THE CORRESPONDING RECOVERIES (%).

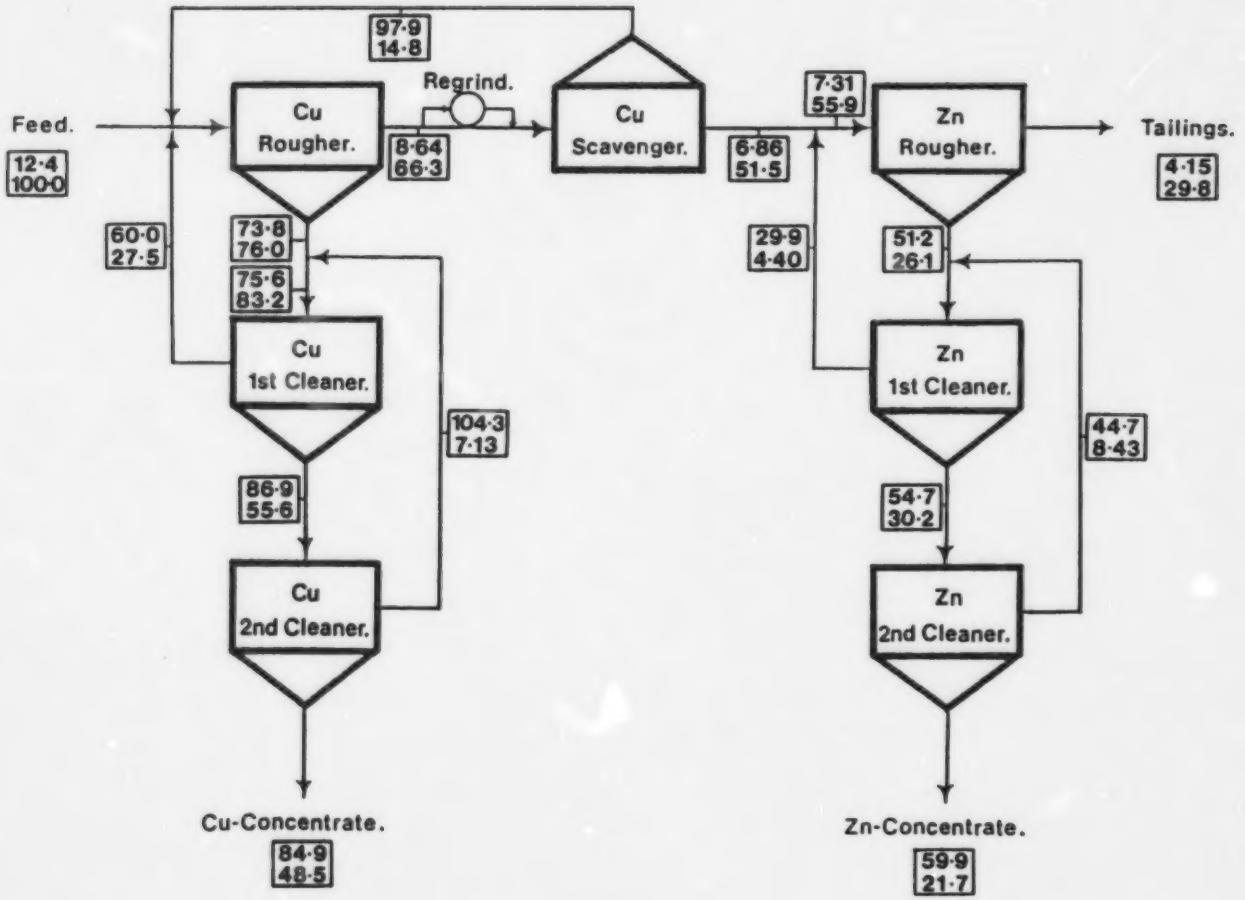

FIGURE 4.
THE TROUT LAKE CONCENTRATOR CIRCUIT DIAGRAM PORTRAYING THE ADJUSTED ASSAYS AND RECOVERIES FOR Ag IN EACH PRODUCT. VALUES IN THE BOXES ARE (TOP) MATERIAL BALANCE ADJUSTED ASSAYS (OZ/TON), AND (BOTTOM) THE CORRESPONDING RECOVERIES (%).

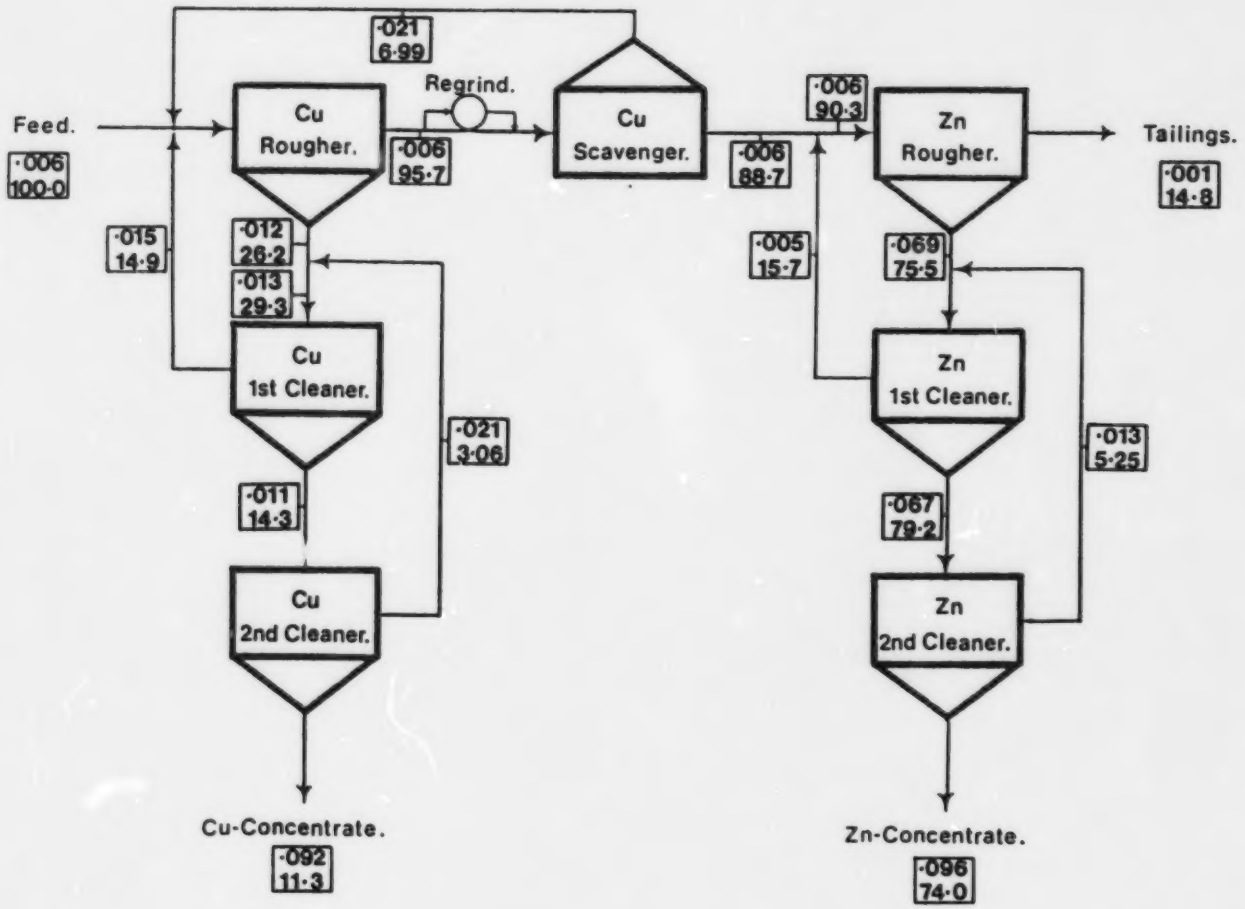

FIGURE 5.
THE TROUT LAKE CONCENTRATOR CIRCUIT DIAGRAM PORTRAYING THE ADJUSTED ASSAYS AND RECOVERIES FOR Cd IN EACH PRODUCT. VALUES IN THE BOXES ARE (TOP) MATERIAL BALANCE ADJUSTED ASSAYS (WT %), AND (BOTTOM) THE CORRESPONDING RECOVERIES (%).

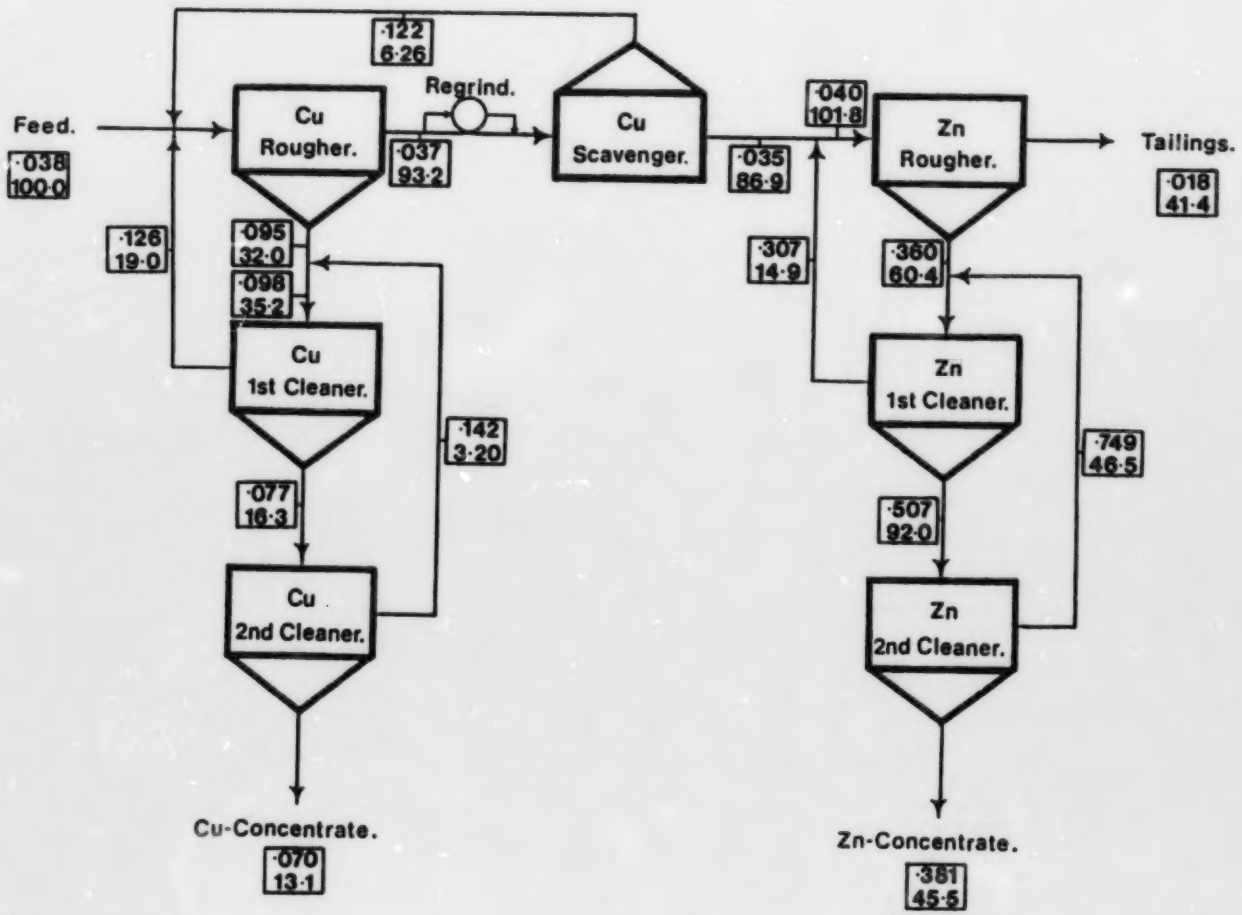

FIGURE 6.
THE TROUT LAKE CONCENTRATOR CIRCUIT DIAGRAM PORTRAYING THE ADJUSTED ASSAYS AND RECOVERIES FOR Pb IN EACH PRODUCT. VALUES IN THE BOXES ARE (TOP) MATERIAL BALANCE ADJUSTED ASSAYS (WT %), AND (BOTTOM) THE CORRESPONDING RECOVERIES (%).

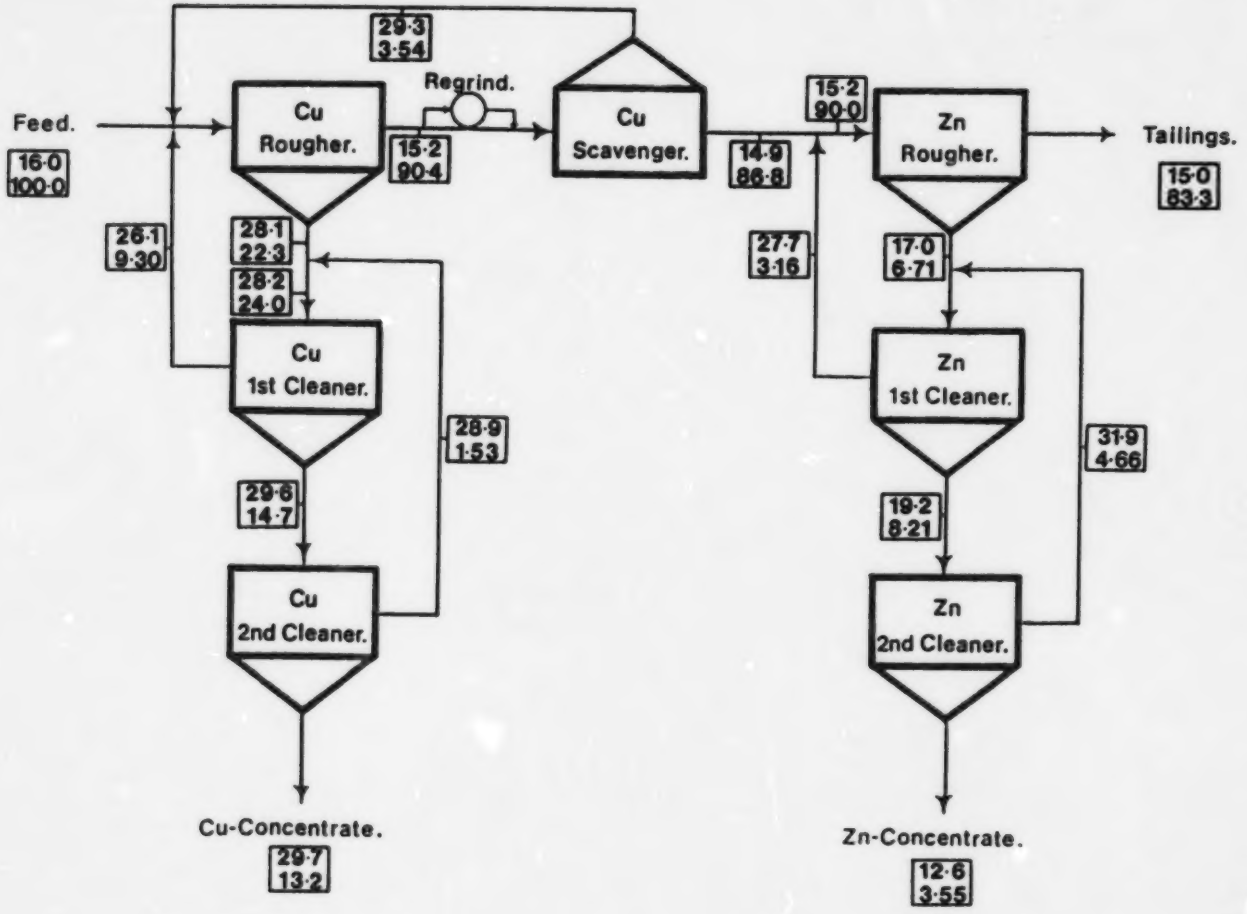

FIGURE 7.
THE TROUT LAKE CONCENTRATOR CIRCUIT DIAGRAM PORTRAYING THE ADJUSTED ASSAYS AND RECOVERIES FOR Fe IN EACH PRODUCT. VALUES IN THE BOXES ARE (TOP) MATERIAL BALANCE ADJUSTED ASSAYS (WT %), AND (BOTTOM) THE CORRESPONDING RECOVERIES (%).

Appendix B

TABLE 1

RAW MINERAL LIBERATIONS (% FREE) IN THE 25 TROUT LAKE MILL PRODUCT SAMPLES.

SAMPLE.	CHALCOPYRITE	SPHALERITE	GALENA
TL.MP-1	56.4	41.8	7.34
TL.MP-8	18.4	14.4	8.57
TL.MP-2	62.5	43.3	----
TL.MP-4	59.7	36.0	10.1
TL.MP-6	58.9	39.7	31.4
TL.MP-15	64.1	34.0	31.0
TL.MP-10	65.7	44.9	19.4
TL.MP-12	75.4	49.7	18.3
TL.MP-21	3.40	57.4	5.45
TL.MP-14	18.3	38.9	18.1
TL.MP-16	17.2	69.1	----
TL.MP-18	11.4	48.1	26.3
TL.MP-20	13.1	74.3	----
TL.MP-22	18.1	63.1	36.2
TL.MP-24	13.5	61.8	32.3
TL.MP-26	0.30	63.3	12.4
TL.MP-3	27.0	43.0	5.70
TL.MP-7	40.4	59.7	4.61
TL.MP-9	3.40	41.0	7.59
TL.MP-11	40.8	25.0	12.9
TL.MP-25	46.8	48.2	----
TL.MP-13	68.7	65.6	9.63
TL.MP-23	11.5	21.7	34.8
TL.MP-19	7.30	12.3	37.6
TL.MP-17	3.90	0.00	4.92

Note : '----' denotes not analysed for.

TABLE 2

RAW % UNLIBERATED MINERAL IN THE 25 TROUT LAKE MILL PRODUCT SAMPLES.

SAMPLE.	CHALCOPYRITE	SPHALERITE	GALENA
TL.MP-1	43.6	58.2	92.7
TL.MP-8	81.6	85.6	91.4
TL.MP-2	37.5	56.4	----
TL.MP-4	40.3	64.0	89.9
TL.MP-6	41.1	60.3	68.6
TL.MP-15	35.9	66.0	69.0
TL.MP-10	34.3	55.1	80.6
TL.MP-12	24.6	50.3	81.7
TL.MP-21	96.6	42.6	94.5
TL.MP-14	81.7	61.1	81.9
TL.MP-16	82.8	30.9	----
TL.MP-18	88.6	51.9	73.7
TL.MP-20	86.9	25.7	----
TL.MP-22	81.9	36.9	63.8
TL.MP-24	86.5	38.2	67.7
TL.MP-26	99.7	36.7	87.6
TL.MP-3	73.0	57.0	94.3
TL.MP-7	59.6	40.3	95.4
TL.MP-9	96.6	59.0	92.4
TL.MP-11	59.2	75.0	87.1
TL.MP-25	53.2	51.8	----
TL.MP-13	31.3	34.4	90.4
TL.MP-23	88.5	78.3	65.2
TL.MP-19	92.7	87.7	62.4
TL.MP-17	96.1	100.0	95.1

Note : '----' denotes not analysed for.

TABLE 3

COMBINED TOTALS OF THE ADJUSTED % FREE AND % UNLIBERATED MINERAL IN THE 17 TROUT LAKE MILL PRODUCTS.

(Combined respective columns from Tables 20 and 24).

PRODUCT.	CHALCOPYRITE	SPHALERITE	GALENA
1. Feed.	96.4	96.6	97.6
2. Cu-Scav Co.	97.4	101.8	100.9
3. Cu-Roug Co.	97.3	99.2	95.7
4. Cu-Cl.1 Fd.	97.5	99.0	96.1
5. Cu-Cl.1 Co.	96.9	98.0	99.2
6. Cu.Cl.2 Co.	96.8	98.2	99.3
7. Zn-Roug Fd.	92.7	96.5	95.8
8. Zn-Roug Co.	90.7	96.0	89.2
9. Zn-Cl.1 Co.	93.1	96.1	96.9
10. Zn-Cl.2 Co.	88.6	95.9	90.4
11. Cu-Roug Tl.	94.6	96.9	97.3
12. Cu-Scav Tl.	92.1	96.5	97.5
13. Cu-Cl.1 Tl.	101.6	99.9	93.4
14. Cu-Cl.2 Tl.	100.1	97.2	98.9
15. Zn-Cl.1 Tl.	100.5	99.9	85.4
16. Zn-Cl.2 Tl.	99.9	98.5	103.2
17. Zn-Roug Tl.	94.5	104.4	104.5

TABLE 4

RAW LIBERATION AND CLASSIFICATION DATA FOR CHALCOPYRITE IN THE 25 MILL PRODUCT SAMPLES.

SAMPLE.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.									% FREE
		0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	43.6	0.84	0.31	1.74	0.71	0.95	3.09	6.10	3.02	9.26	17.4	56.4
8. Cu-Scav Co.	81.6	3.54	4.34	4.07	1.27	2.11	3.18	0.85	2.31	2.04	58.1	18.4
2. Cu-Roug Co.	37.5	0.75	0.48	1.92	1.55	2.26	1.15	4.16	3.44	7.76	14.0	62.5
4. Cu-Roug Co.	40.3	0.58	0.38	0.63	0.44	0.47	0.26	0.41	0.85	2.06	34.2	59.7
6. Cu-Roug Co.	41.1	0.57	0.53	0.63	0.86	0.07	0.57	0.72	0.80	1.58	34.8	58.9
15. Cu-Cl.1 Fd.	35.9	0.73	0.75	1.16	1.15	1.16	0.28	0.55	0.72	1.41	27.9	64.1
10. Cu-Cl.1 Co.	34.3	0.29	0.65	0.49	0.79	0.21	0.49	0.61	0.77	2.03	27.9	65.7
12. Cu-Cl.2 Co.	24.6	0.10	0.19	0.18	0.14	0.32	0.30	0.42	0.55	1.78	20.7	75.4
21. In-Roug Fd.	96.6	57.4	1.85	16.7	1.79	0.00	0.00	0.00	0.00	0.00	19.0	3.40
14. In-Roug Co.	82.7	20.2	11.7	11.3	3.96	6.21	3.33	3.99	7.75	3.66	10.8	18.3
16. In-Roug Co.	82.8	9.03	8.06	3.18	18.7	0.64	6.62	0.00	0.00	0.00	36.6	17.2
18. In-Roug Co.	88.6	12.9	17.8	5.66	2.19	3.93	2.81	2.75	1.31	5.49	34.1	11.4
20. In-Roug Co.	86.9	11.4	16.5	5.98	0.86	6.22	7.35	0.20	0.33	1.03	37.4	13.1
22. In-Roug Co.	81.9	26.4	21.1	11.1	3.96	3.51	0.24	0.50	0.16	4.16	11.0	18.1
24. In-Cl.1 Co.	86.5	20.8	8.61	4.54	9.62	1.93	12.9	3.36	0.40	4.40	19.9	13.5
26. In-Cl.2 Co.	99.7	18.5	24.5	16.4	4.23	8.73	11.9	2.45	1.56	11.5	0.00	0.30
3. Cu-Roug Tl.	73.0	16.8	14.1	0.28	4.42	33.6	0.25	0.00	0.00	3.48	0.00	27.0
7. Cu-Scav Tl.	59.6	10.6	2.06	1.51	6.38	5.39	10.5	0.95	12.8	0.40	9.06	40.4
9. Cu-Scav Tl.	96.6	12.1	30.9	0.91	0.00	0.18	5.72	0.00	35.1	0.00	11.8	3.4
11. Cu-Cl.1 Tl.	59.2	4.32	8.58	7.05	3.82	4.57	1.63	1.50	1.66	1.79	24.3	40.8
25. Cu-Cl.1 Tl.	53.2	4.34	4.18	4.40	2.86	1.44	1.81	5.28	1.68	1.69	29.3	46.8
13. Cu-Cl.2 Tl.	31.3	1.24	1.32	1.32	1.22	1.74	1.91	0.82	1.37	1.11	19.3	68.7
23. In-Cl.1 Tl.	88.5	17.0	8.77	5.77	1.74	21.8	1.64	1.55	0.00	0.00	30.3	11.5
19. In-Cl.2 Tl.	92.7	9.79	10.4	8.32	1.01	3.70	0.21	1.09	12.8	5.54	39.8	7.30
17. In-Roug Tl.	96.1	26.7	2.66	0.00	0.00	25.5	0.00	0.00	0.00	41.5	0.00	3.90

TABLE 5

RAW MODAL FRACTION OF THE ADJUSTED Cu ASSAY OCCURRING AS FREE GRAINS, AND IN EACH CLASS OF UNLIBERATED CHALCOPYRITE IN THE 17 MILL PRODUCTS.

PRODUCT.	RAW % FREE	RAW % LOCKED	ADJUSTED LOCKED MODAL FRACTION	MODAL FRACTION OF Cu ASSAY IN THE CLASSIFICATION OF UNLIBERATED GRAINS										ADJUSTED FREE MODAL FRACTION
				0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	56.4	43.6	0.94	0.02	0.01	0.04	0.02	0.02	0.07	0.13	0.07	0.20	0.38	1.22
2. Cu-Scav Co.	18.4	81.6	6.82	0.30	0.36	0.34	0.11	0.18	0.27	0.07	0.19	0.17	4.86	1.54
3. Cu-Roug Co.	60.4	39.6	6.89	0.11	0.08	0.18	0.17	0.16	0.12	0.31	0.30	0.66	4.82	10.5
4. Cu-Cl.1 Fd.	64.1	35.9	6.15	0.13	0.13	0.20	0.20	0.20	0.05	0.09	0.12	0.24	4.78	11.0
5. Cu-Cl.1 Co.	65.7	34.3	9.02	0.08	0.17	0.13	0.21	0.06	0.13	0.16	0.20	0.53	7.34	17.3
6. Cu-Cl.2 Co.	75.4	24.6	6.89	0.03	0.05	0.05	0.04	0.09	0.08	0.12	0.15	0.50	5.80	21.1
7. Zn-Roug Fd.	3.4	96.6	0.20	0.12	0.00	0.04	0.00	0.00	0.00	0.00	0.00	0.00	0.04	0.01
8. Zn-Roug Co.	15.6	84.4	1.22	0.23	0.22	0.11	0.09	0.06	0.06	0.02	0.03	0.04	0.38	0.23
9. Zn-Cl.1 Co.	13.5	86.5	1.60	0.39	0.16	0.08	0.18	0.04	0.24	0.06	0.01	0.08	0.37	0.25
10. Zn-Cl.2 Co.	0.3	99.7	1.66	0.31	0.41	0.27	0.07	0.15	0.20	0.04	0.03	0.19	0.00	0.01
11. Cu-Roug Tl.	27.0	73.0	0.26	0.06	0.05	0.00	0.02	0.12	0.00	0.00	0.00	0.01	0.00	0.10
12. Cu-Scav Tl.	21.9	78.1	0.15	0.02	0.03	0.00	0.01	0.01	0.02	0.00	0.05	0.00	0.02	0.04
13. Cu-Cl.1 Tl.	43.8	56.2	2.45	0.19	0.28	0.25	0.15	0.13	0.08	0.15	0.07	0.08	1.17	1.91
14. Cu-Cl.2 Tl.	68.7	31.3	3.75	0.15	0.16	0.16	0.15	0.21	0.23	0.10	0.16	0.13	2.31	8.24
15. Zn-Cl.1 Tl.	11.5	88.5	0.80	0.15	0.08	0.05	0.02	0.20	0.02	0.01	0.00	0.00	0.27	0.10
16. Zn-Cl.2 Tl.	7.3	92.7	2.03	0.21	0.23	0.18	0.02	0.08	0.01	0.02	0.28	0.12	0.87	0.16
17. Zn-Roug Tl.	3.9	96.1	0.11	0.03	0.00	0.00	0.00	0.03	0.00	0.00	0.00	0.05	0.00	0.01

TABLE 6

ADJUSTED MODAL FRACTION OF THE Cu ASSAY OCCURRING AS FREE GRAINS, AND IN COMPOSITE CLASSES OF UNLIBERATED CHALCOPYRITE IN THE 17 MILL PRODUCTS.

PRODUCT.	ADJUSTED Cu ASSAY	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	90-100	70-100
1. Feed	2.16	1.31	0.77	0.07	0.03	0.15	0.61	1.79	1.95
2. Cu-Scav Co.	8.36	1.80	6.35	1.04	0.68	1.72	5.22	5.11	4.26
3. Cu-Roug Co.	17.5	11.0	5.95	0.40	0.49	0.93	4.99	15.3	16.3
4. Cu-Cl.1 Fd.	17.1	10.9	5.82	0.41	0.51	0.95	4.84	15.1	16.0
5. Cu-Cl.1 Co.	26.3	17.3	8.18	0.19	0.46	0.64	7.35	23.6	25.1
6. Cu-Cl.2 Co.	28.0	18.4	8.71	0.15	0.43	0.57	7.91	25.2	26.8
7. Zn-Roug Fd.	0.21	0.01	0.18	0.07	0.00	0.13	0.06	0.02	0.06
8. Zn-Roug Co.	1.45	0.04	1.27	0.61	0.20	0.99	0.24	0.14	0.29
9. Zn-Cl.1 Co.	1.85	0.07	1.65	0.70	0.19	1.04	0.52	0.42	0.67
10. Zn-Cl.2 Co.	1.67	0.01	1.47	0.75	0.21	1.19	0.22	0.01	0.26
11. Cu-Roug Tl.	0.36	0.05	0.29	0.08	0.01	0.15	0.16	0.12	0.14
12. Cu-Scav Tl.	0.19	0.01	0.17	0.06	0.00	0.12	0.06	0.01	0.06
13. Cu-Cl.1 Tl.	4.36	1.90	2.53	0.72	0.58	1.37	1.35	3.09	3.23
14. Cu-Cl.2 Tl.	12.0	8.25	3.77	0.51	0.71	1.21	2.61	10.6	10.8
15. Zn-Cl.1 Tl.	0.90	0.11	0.80	0.28	0.16	0.51	0.27	0.45	0.39
16. Zn-Cl.2 Tl.	2.19	0.19	2.00	0.60	0.14	0.77	1.11	1.22	1.48
17. Zn-Roug Tl.	0.12	0.01	0.10	0.03	-0.01	0.06	0.05	0.01	0.05

TABLE 7

RAW LIBERATION DATA FOR SELECTED CATEGORIES OF UNLIBERATED CHALCOPYRITE IN THE 25 TROUT LAKE MILL PRODUCT SAMPLES.

(in Area % Chalcopyrite in each Category).

SAMPLE.	FREE GRAINS (% Free)	UNLIBERATED GRAINS				
		All.	Gang.	Sulf.	Py+Po.	Sphal.
1. Feed.	56.4	43.6	37.0	17.3	14.3	12.3
8. Cu-Scav Co.	18.4	81.6	42.8	68.5	47.8	45.0
2. Cu-Roug Co.	62.5	37.5	14.3	26.3	19.9	9.7
4. Cu-Roug Co.	59.7	40.3	19.4	30.3	23.8	9.4
6. Cu-Roug Co.	58.9	41.1	26.7	22.9	14.2	8.0
15. Cu-Cl.1 Fd.	64.1	35.9	15.1	24.5	14.7	37.8
10. Cu-Cl.1 Co.	65.7	34.3	15.8	26.4	12.1	16.1
12. Cu-Cl.2 Co.	75.4	24.6	17.8	15.4	10.0	5.7
21. Zn-Roug Fd.	3.4	96.6	----	----	----	----
14. Zn-Roug Co.	18.3	81.7	13.5	74.5	32.0	55.2
16. Zn-Roug Co.	17.2	82.8	30.3	82.8	74.6	37.9
18. Zn-Roug Co.	11.4	88.6	35.5	80.1	47.6	44.4
20. Zn-Roug Co.	13.1	86.9	42.8	60.3	42.2	37.0
22. Zn-Roug Co.	18.1	81.9	29.4	70.3	28.6	46.2
24. Zn-Cl.1 Co.	13.5	86.5	36.6	71.3	45.8	43.0
26. Zn-Cl.2 Co.	0.3	99.7	41.4	99.7	68.0	98.2
3. Cu-Roug Tl.	27.0	73.0	63.7	36.5	24.8	30.4
7. Cu-Scav Tl.	40.4	59.6	44.9	64.1	60.1	44.6
9. Cu-Scav Tl.	3.4	96.6	63.5	88.5	59.2	83.1
11. Cu-Cl.1 Tl.	40.8	59.2	19.0	46.6	29.9	20.4
25. Cu-Cl.1 Tl.	46.8	53.2	17.4	43.5	28.3	20.8
13. Cu-Cl.2 Tl.	68.7	31.3	11.4	23.0	16.3	6.3
23. Zn-Cl.1 Tl.	11.5	88.5	78.0	76.0	43.9	51.3
19. Zn-Cl.2 Tl.	7.3	92.7	63.1	79.7	52.7	49.1
17. Zn-Roug Tl.	3.9	96.1	96.1	69.0	69.0	----

Note : '----' denotes data lost during processing.

TABLE 8

RAW LIBERATION AND CLASSIFICATION DATA FOR SPHALERITE IN THE 25 MILL PRODUCT SAMPLES.

SAMPLE.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.									% FREE
		0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	58.2	0.40	0.60	0.10	1.23	1.91	2.30	3.42	4.65	2.22	41.3	41.8
8. Cu-Scav Co.	85.6	1.79	1.10	1.31	3.10	2.62	1.79	1.57	7.06	14.2	50.8	14.4
2. Cu-Roug Co.	56.7	1.64	1.21	1.50	2.52	3.73	4.54	5.60	2.44	10.6	22.9	43.3
4. Cu-Roug Co.	64.0	1.73	2.12	1.34	2.29	1.21	3.04	2.48	10.2	9.09	30.5	36.0
6. Cu-Roug Co.	60.3	1.33	1.46	0.64	3.86	0.83	2.41	8.50	1.81	5.99	33.7	39.7
15. Cu-Cl.1 Fd.	66.0	0.74	0.82	1.39	1.47	1.30	6.02	7.05	6.96	14.1	26.4	34.0
10. Cu-Cl.1 Co.	55.1	3.27	1.49	0.61	2.06	1.61	0.37	2.49	4.52	7.35	31.3	44.9
12. Cu-Cl.2 Co.	50.3	1.34	1.35	1.29	1.25	2.60	1.99	1.86	2.73	9.02	26.8	49.7
21. Zn-Roug Fd.	42.6	1.41	0.17	0.78	0.34	1.04	2.32	0.27	3.63	1.22	31.3	57.4
14. Zn-Roug Co.	61.1	0.32	0.50	1.41	0.44	1.12	1.00	1.69	2.61	8.73	43.3	38.9
16. Zn-Roug Co.	30.9	0.26	0.52	1.04	0.13	1.41	0.59	1.00	1.51	4.14	20.3	69.1
18. Zn-Roug Co.	51.9	0.93	1.39	1.05	1.27	1.17	2.00	1.26	3.19	9.19	30.5	48.1
20. Zn-Roug Co.	25.7	0.19	0.28	0.58	0.31	1.69	0.78	1.79	0.72	2.65	17.7	74.3
22. Zn-Roug Co.	36.9	0.35	0.68	0.86	0.76	1.07	1.02	0.99	1.73	2.37	27.1	63.1
24. Zn-Cl.1 Co.	38.2	0.73	1.04	0.76	0.39	1.34	1.53	0.51	1.26	2.70	27.9	61.8
26. Zn-Cl.2 Co.	36.8	0.17	0.48	0.55	0.83	0.72	2.20	0.70	1.55	5.42	24.3	63.2
5. Cu-Roug Tl.	57.0	0.39	2.60	0.85	0.40	0.00	0.22	2.61	5.11	14.6	30.2	43.0
7. Cu-Scav Tl.	40.3	0.71	1.49	0.92	0.63	0.19	0.44	2.75	0.35	7.21	25.6	59.7
9. Cu-Scav Tl.	59.0	1.01	1.76	0.93	0.70	3.85	1.86	1.76	5.27	26.2	15.6	41.0
11. Cu-Cl.1 Tl.	75.0	1.16	1.85	1.46	1.22	1.46	1.46	2.90	12.2	10.4	41.0	25.0
25. Cu-Cl.1 Tl.	51.8	0.61	1.16	0.52	0.51	1.10	1.56	3.81	6.06	10.3	26.2	48.2
13. Cu-Cl.2 Tl.	34.4	0.45	0.51	0.29	0.88	1.70	1.38	2.81	3.13	6.66	16.6	65.6
23. Zn-Cl.1 Tl.	78.3	27.7	19.5	6.59	6.34	2.29	2.01	1.80	0.15	8.92	3.16	21.7
19. Zn-Cl.2 Tl.	87.7	13.5	20.1	11.2	9.51	9.32	13.5	5.16	2.18	1.33	1.97	12.3
17. Zn-Roug Tl.	100.0	31.3	19.0	0.00	0.00	0.00	0.00	5.66	0.00	44.2	0.00	0.00

TABLE 9

RAW MODAL FRACTION OF THE ADJUSTED Zn ASSAY OCCURRING AS FREE GRAINS, AND IN EACH CLASS OF UNLIBERATED SPHALERITE IN THE 17 MILL PRODUCTS.

PRODUCT.	RAW % FREE	RAW % LOCKED	ADJUSTED LOCKED MODAL FRACTION	MODAL FRACTION OF Zn ASSAY IN THE CLASSIFICATION OF UNLIBERATED GRAINS										ADJUSTED FREE MODAL FRACTION
				0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	41.8	58.2	1.50	0.01	0.02	0.00	0.03	0.05	0.06	0.09	0.12	0.06	1.06	1.07
2. Cu-Scav Co.	14.4	85.6	9.52	0.20	0.12	0.15	0.35	0.29	0.20	0.18	0.79	1.58	5.65	1.60
3. Cu-Roug Co.	39.7	60.3	3.26	0.08	0.09	0.06	0.15	0.10	0.18	0.30	0.26	0.46	1.57	2.15
4. Cu-Cl.1 Fd.	34.0	66.0	3.75	0.04	0.05	0.08	0.08	0.07	0.34	0.40	0.40	0.80	1.50	1.93
5. Cu-Cl.1 Co.	44.9	55.1	2.54	0.15	0.07	0.03	0.10	0.07	0.02	0.12	0.21	0.34	1.44	2.07
6. Cu-Cl.2 Co.	49.7	50.3	2.01	0.05	0.05	0.05	0.05	0.10	0.08	0.07	0.11	0.36	1.07	1.98
7. Zn-Roug Fd.	57.4	42.6	1.05	0.04	0.00	0.02	0.01	0.03	0.06	0.01	0.09	0.03	0.77	1.41
8. Zn-Roug Co.	58.7	41.3	14.2	0.14	0.23	0.34	0.20	0.37	0.37	0.46	0.67	1.86	9.55	20.2
9. Zn-Cl.1 Co.	61.8	38.2	12.6	0.24	0.34	0.25	0.13	0.44	0.51	0.17	0.42	0.89	9.22	20.4
10. Zn-Cl.2 Co.	63.2	36.8	17.4	0.08	0.23	0.26	0.39	0.34	1.04	0.33	0.73	2.56	11.5	29.9
11. Cu-Roug Tl.	43.0	57.0	1.51	0.01	0.07	0.02	0.01	0.00	0.01	0.07	0.14	0.39	0.80	1.14
12. Cu-Scav Tl.	50.4	49.6	1.22	0.02	0.04	0.02	0.02	0.05	0.03	0.06	0.07	0.41	0.51	1.24
13. Cu-Cl.1 Tl.	36.6	63.4	4.55	0.16	0.11	0.07	0.06	0.09	0.11	0.24	0.66	0.75	2.41	2.63
14. Cu-Cl.2 Tl.	65.6	34.4	3.32	0.04	0.05	0.03	0.09	0.16	0.13	0.27	0.30	0.64	1.60	6.34
15. Zn-Cl.1 Tl.	21.7	78.3	1.88	0.67	0.47	0.16	0.15	0.06	0.05	0.04	0.00	0.21	0.08	0.52
16. Zn-Cl.2 Tl.	12.3	87.7	4.91	0.76	1.13	0.63	0.53	0.52	0.76	0.29	0.12	0.08	0.11	0.69
17. Zn-Roug Tl.	0.00	100.0	0.19	0.06	0.04	0.00	0.00	0.00	0.01	0.00	0.08	0.00	0.00	0.00

TABLE 10

ADJUSTED MODAL FRACTION OF THE Zn ASSAY OCCURRING AS FREE GRAINS, AND IN COMPOSITE CLASSES OF UNLIBERATED SPHALERITE IN THE 17 MILL PRODUCTS.

PRODUCT.	ADJUSTED Zn ASSAY	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING.			COMPOSITE CLASSES OF FREE AND SUB-FREE.		
				0-30	30-70	0-70	70-99.9	90-100	70-100
1. Feed	2.57	1.30	1.19	0.05	0.12	0.22	0.95	2.05	2.35
2. Cu-Scav Co.	11.1	1.60	9.70	0.49	0.95	1.46	8.22	7.21	9.64
3. Cu-Roug Co.	5.41	2.03	3.34	0.19	0.45	0.69	2.54	3.68	4.61
4. Cu-Cl.1 Fd.	5.68	2.28	3.34	0.19	0.47	0.69	2.54	3.93	4.87
5. Cu-Cl.1 Co.	4.60	2.17	2.34	0.15	0.36	0.56	1.76	3.44	4.00
6. Cu-Cl.2 Co.	3.99	1.71	2.21	0.15	0.32	0.53	1.66	2.92	3.42
7. Zn-Roug Fd.	2.46	1.25	1.12	0.06	0.11	0.22	0.90	1.95	2.24
8. Zn-Roug Co.	34.3	18.8	14.1	0.67	1.30	1.89	12.4	29.4	32.6
9. Zn-Cl.1 Co.	33.0	17.5	14.2	0.98	1.69	2.61	11.5	27.3	30.3
10. Zn-Cl.2 Co.	47.3	26.3	19.0	0.41	1.70	2.02	17.3	41.0	45.5
11. Cu-Roug Tl.	2.64	1.27	1.28	0.05	0.12	0.22	1.06	2.09	2.42
12. Cu-Scav Tl.	2.46	1.26	1.11	0.04	0.10	0.19	0.91	1.98	2.27
13. Cu-Cl.1 Tl.	7.18	2.44	4.74	0.24	0.61	0.89	3.64	4.62	6.09
14. Cu-Cl.2 Tl.	9.66	6.01	3.38	0.12	0.64	0.78	2.59	7.81	8.85
15. Zn-Cl.1 Tl.	2.40	0.52	1.88	1.30	0.30	1.56	0.29	0.60	0.81
16. Zn-Cl.2 Tl.	5.60	0.69	4.83	2.06	1.67	3.73	0.31	0.80	1.00
17. Zn-Roug Tl.	0.19	0.00	0.20	0.02	0.02	0.10	0.08	0.00	0.08

TABLE 11

RAW LIBERATION DATA FOR SELECTED CATEGORIES OF UNLIBERATED SPHALERITE IN THE 25 TROUT LAKE MILL PRODUCT SAMPLES.

(in Area % Sphalerite in each Category).

SAMPLE.	FREE GRAINS (% Free)	UNLIBERATED GRAINS			
		All.	Cpy.	Py+Po+Asp.	Sulf.
1. Feed.	41.8	58.2	21.2	35.8	38.6
8. Cu-Scav Co.	14.4	85.6	80.2	22.9	82.3
2. Cu-Roug Co.	43.3	56.7	31.1	38.1	----
4. Cu-Roug Co.	36.0	64.0	57.9	19.3	61.2
6. Cu-Roug Co.	39.7	60.3	54.9	8.9	56.0
15. Cu-Cl.1 Fd.	34.0	66.0	62.5	11.3	63.8
10. Cu-Cl.1 Co.	44.9	55.1	48.8	8.2	49.7
12. Cu-Cl.2 Co.	49.7	50.3	44.2	3.9	44.2
21. Zn-Roug Fd.	57.4	42.6	22.0	30.8	35.2
14. Zn-Roug Co.	38.9	61.1	53.1	8.6	58.0
16. Zn-Roug Co.	69.1	30.9	12.1	10.9	23.7
18. Zn-Roug Co.	48.1	51.9	22.8	23.0	39.2
20. Zn-Roug Co.	74.3	25.7	9.3	7.9	16.9
22. Zn-Roug Co.	63.1	36.9	16.4	10.8	25.6
24. Zn-Cl.1 Co.	61.8	38.2	20.4	12.0	31.7
26. Zn-Cl.2 Co.	63.2	36.8	13.9	19.2	32.9
3. Cu-Roug Tl.	43.0	57.0	28.5	34.4	46.2
7. Cu-Scav Tl.	59.7	40.3	23.3	25.9	36.6
9. Cu-Scav Tl.	41.0	59.0	56.1	48.9	56.3
11. Cu-Cl.1 Tl.	25.0	75.0	68.8	10.8	70.8
25. Cu-Cl.1 Tl.	48.2	51.8	42.8	14.5	48.6
13. Cu-Cl.2 Tl.	65.6	34.4	29.0	6.0	32.4
23. Zn-Cl.1 Tl.	21.7	78.3	24.4	54.5	70.8
19. Zn-Cl.2 Tl.	12.3	87.7	11.7	71.5	74.9
17. Zn-Roug Tl.	0.0	100.0	----	----	----

Note : '----' denotes data lost during processing.

TABLE 12

RAW LIBERATION AND CLASSIFICATION DATA FOR GALENA IN THE 25 MILL PRODUCT SAMPLES.

SAMPLE.	% LOCKED	CLASSIFICATION OF UNLIBERATED GRAINS.									% FREE
		0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed (Run 1)	90.8	16.9	1.59	12.9	10.6	0.00	8.17	0.00	4.58	6.03	30.0	9.23
1. Feed (Run 2)	93.2	53.5	1.74	4.55	5.47	0.00	9.45	4.77	0.00	0.44	13.3	6.82
8. Cu-Scav Co.	91.4	50.9	1.52	9.29	0.80	0.00	0.00	1.42	0.00	7.67	19.9	8.57
4. Cu-Roug Co.	89.9	14.8	6.78	8.20	0.65	1.19	7.99	4.50	8.15	24.3	13.4	10.1
6. Cu-Roug Co.	68.6	17.1	4.94	2.20	3.75	2.85	2.33	10.4	2.20	0.48	22.1	31.4
15. Cu-Cl.1 Fd.	69.0	25.5	2.81	5.18	0.00	10.1	0.00	3.92	1.70	0.00	19.8	31.0
10. Cu-Cl.1 Co.	80.6	17.3	15.0	11.5	7.37	3.64	0.60	0.00	1.71	9.26	14.2	19.4
12. Cu-Cl.2 Co. (Run 1)	75.6	14.1	11.0	5.87	4.76	2.19	5.08	0.00	0.78	8.75	23.4	24.4
12. Cu-Cl.2 Co. (Run 2)	87.8	29.5	19.2	2.47	0.00	0.00	12.4	1.48	5.33	17.3	0.00	12.2
21. Zn-Roug Fd.	94.5	7.30	4.13	2.23	0.46	3.75	1.99	2.98	3.41	7.20	61.1	5.45
14. Zn-Roug Co. (Run 1)	80.1	22.3	10.9	1.74	4.46	3.49	5.34	0.00	0.00	2.71	29.1	19.9
14. Zn-Roug Co. (Run 2)	82.2	----	----	----	----	----	----	----	----	----	----	17.8
18. Zn-Roug Co.	73.7	14.5	3.26	1.78	6.84	1.83	0.29	1.96	9.27	5.60	29.0	26.3
22. Zn-Roug Co.	63.8	17.0	8.30	4.75	0.21	3.98	2.67	2.55	3.21	3.03	18.0	36.2
24. Zn-Cl.1 Co.	67.7	9.03	6.66	1.49	1.49	0.06	0.75	0.40	6.11	4.18	37.8	32.3
26. Zn-Cl.2 Co.	87.6	13.8	4.69	5.31	0.55	1.99	6.96	1.33	5.07	17.6	30.2	12.4
3. Cu-Roug Tl.	94.3	39.9	14.6	5.97	0.84	3.78	10.6	1.79	0.28	1.00	15.9	5.70
7. Cu-Scav Tl.	95.4	6.01	1.98	0.12	0.42	1.09	0.00	0.00	6.85	12.2	67.1	4.61
9. Cu-Scav Tl.	92.4	9.54	2.67	2.82	5.64	1.68	2.85	6.39	2.09	4.94	54.1	7.59
11. Cu-Cl.1 Tl. (Run 1)	77.2	7.56	3.76	2.18	3.35	1.66	1.66	7.31	6.99	12.6	30.3	22.8
11. Cu-Cl.1 Tl. (Run 2)	97.0	14.8	69.9	0.00	0.00	0.00	0.00	0.00	2.65	1.87	8.02	2.97
13. Cu-Cl.2 Tl.	90.4	11.1	9.52	2.83	5.60	7.93	4.96	3.30	2.54	12.5	30.1	9.63
23. Zn-Cl.1 Tl.	65.2	4.06	6.68	0.63	2.09	0.86	0.93	4.95	0.74	6.67	37.6	34.8
19. Zn-Cl.2 Tl.	62.4	0.74	1.48	0.74	1.01	2.78	0.64	1.95	1.88	6.12	45.3	37.6
17. Zn-Roug Tl. (Run 1)	91.1	3.57	0.00	0.24	0.21	0.22	0.00	0.00	2.67	9.11	75.5	8.94
17. Zn-Roug Tl. (Run 2)	99.1	2.18	18.8	0.11	0.17	0.05	0.95	2.26	1.69	3.30	69.6	0.89

TABLE 13

RAW MODAL FRACTION OF THE ADJUSTED Pb ASSAY OCCURRING AS FREE GRAINS, AND IN EACH CLASS OF UNLIBERATED GALENA IN THE 17 MILL PRODUCTS.

PRODUCT.	RAW % FREE	RAW % LOCKED	ADJUSTED LOCKED MODAL FRACTION	MODAL FRACTION OF Pb ASSAY IN THE CLASSIFICATION OF UNLIBERATED GRAINS										ADJUSTED FREE MODAL FRACTION
				0-10	10-20	20-30	30-40	40-50	50-60	60-70	70-80	80-90	90-99	
1. Feed	8.25	91.8	0.035	0.012	0.001	0.004	0.003	0.000	0.003	0.001	0.001	0.002	0.009	0.003
2. Cu-Scav Co.	8.57	91.4	0.112	0.062	0.002	0.011	0.001	0.000	0.000	0.002	0.000	0.009	0.024	0.011
3. Cu-Roug Co.	20.8	79.2	0.075	0.015	0.006	0.005	0.002	0.002	0.005	0.007	0.005	0.012	0.017	0.020
4. Cu-Cl.1 Fd.	31.0	69.0	0.068	0.025	0.003	0.005	0.000	0.010	0.000	0.004	0.002	0.000	0.019	0.030
5. Cu-Cl.1 Co.	19.4	80.6	0.062	0.013	0.012	0.009	0.006	0.003	0.001	0.000	0.001	0.007	0.011	0.015
6. Cu-Cl.2 Co.	18.3	81.7	0.057	0.015	0.011	0.003	0.002	0.001	0.006	0.001	0.002	0.009	0.008	0.013
7. Zn-Roug Fd.	5.45	94.5	0.038	0.003	0.002	0.001	0.000	0.002	0.001	0.001	0.001	0.003	0.024	0.002
8. Zn-Roug Co.	27.4	72.6	0.261	0.064	0.027	0.010	0.014	0.011	0.010	0.005	0.015	0.014	0.091	0.099
9. Zn-Cl.1 Co.	32.3	67.7	0.343	0.046	0.034	0.008	0.008	0.000	0.004	0.002	0.031	0.021	0.192	0.164
10. Zn-Cl.2 Co.	12.4	87.6	0.334	0.053	0.018	0.020	0.002	0.008	0.027	0.005	0.019	0.067	0.115	0.047
11. Cu-Roug Tl.	5.70	94.3	0.035	0.015	0.005	0.002	0.000	0.001	0.004	0.001	0.000	0.000	0.006	0.002
12. Cu-Scav Tl.	6.10	93.9	0.033	0.003	0.001	0.001	0.001	0.001	0.001	0.001	0.002	0.003	0.021	0.002
13. Cu-Cl.1 Tl.	11.6	88.4	0.111	0.015	0.052	0.001	0.002	0.001	0.001	0.004	0.006	0.008	0.022	0.015
14. Cu-Cl.2 Tl.	9.63	90.4	0.128	0.016	0.014	0.004	0.008	0.011	0.007	0.005	0.004	0.018	0.043	0.014
15. Zn-Cl.1 Tl.	34.8	65.2	0.200	0.013	0.021	0.002	0.006	0.003	0.003	0.015	0.002	0.021	0.115	0.107
16. Zn-Cl.2 Tl.	37.6	62.4	0.467	0.006	0.011	0.006	0.008	0.021	0.003	0.015	0.014	0.046	0.339	0.282
17. Zn-Roug Tl.	4.92	95.1	0.017	0.001	0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.001	0.013	0.001

TABLE 14

ADJUSTED MODAL FRACTION OF THE Pb ASSAY OCCURRING AS FREE GRAINS, AND IN COMPOSITE CLASSES OF UNLIBERATED GALENA IN THE 17 MILL PRODUCTS.

PRODUCT.	ADJUSTED Pb ASSAY	% FREE (99.9-100)	% LOCKED (0-99.9)	COMPOSITE CLASSES OF MIDDLING AND SUB-MIDDLING. 0-30	30-70	0-70	COMPOSITE CLASSES OF FREE AND SUB-FREE. 70-99.9	90-100	70-100
1. Feed	0.038	0.003	0.034	0.010	0.004	0.013	0.010	0.013	0.014
2. Cu-Scav Co.	0.122	0.011	0.112	0.078	0.003	0.080	0.031	0.033	0.041
3. Cu-Roug Co.	0.095	0.016	0.075	0.034	0.010	0.049	0.024	0.032	0.044
4. Cu-Cl.1 Fd.	0.098	0.016	0.078	0.034	0.011	0.050	0.027	0.033	0.046
5. Cu-Cl.1 Co.	0.078	0.015	0.062	0.029	0.012	0.041	0.021	0.027	0.038
6. Cu-Cl.2 Co.	0.070	0.015	0.054	0.028	0.010	0.038	0.017	0.023	0.033
7. Zn-Roug Fd.	0.040	0.003	0.036	0.010	0.004	0.013	0.012	0.017	0.018
8. Zn-Roug Co.	0.360	0.038	0.282	0.099	0.039	0.138	0.121	0.163	0.189
9. Zn-Cl.1 Co.	0.506	0.127	0.364	0.089	0.038	0.132	0.218	0.313	0.343
10. Zn-Cl.2 Co.	0.380	0.027	0.317	0.124	0.041	0.166	0.111	0.129	0.156
11. Cu-Roug Tl.	0.037	0.002	0.034	0.011	0.003	0.013	0.010	0.013	0.013
12. Cu-Scav Tl.	0.035	0.002	0.032	0.010	0.003	0.012	0.010	0.012	0.013
13. Cu-Cl.1 Tl.	0.126	0.017	0.100	0.042	0.009	0.062	0.034	0.043	0.057
14. Cu-Cl.2 Tl.	0.142	0.014	0.126	0.035	0.029	0.065	0.059	0.058	0.078
15. Zn-Cl.1 Tl.	0.307	0.064	0.198	0.036	0.032	0.069	0.145	0.246	0.272
16. Zn-Cl.2 Tl.	0.749	0.320	0.454	0.023	0.031	0.065	0.423	0.666	0.733
17. Zn-Roug Tl.	0.018	0.001	0.018	0.004	0.001	0.004	0.004	0.006	0.006

Appendix C

TABLE 1

THE GRAIN SIZE DISTRIBUTION OF FREE CHALCOPYRITE IN THE 25 MILL PRODUCTS.

	Sample.	Classes (um). -3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	Area. (um^2)
1	Float Feed.	0.07	2.12	8.81	18.2	20.9	12.5	19.8	11.4	6.28	0.00	0.00	0.00	0.00	34600.
8	Cu Scav Co.	0.49	1.12	1.56	2.39	5.15	4.36	7.79	6.98	26.7	18.9	24.5	0.00	0.00	66700.
2	Cu Roug Co.	0.03	0.17	4.24	7.61	12.8	17.9	20.2	10.9	13.0	11.8	0.00	0.00	0.00	262000.
4	Cu Roug Co.	0.08	1.03	3.27	5.25	8.60	11.7	14.2	20.5	16.5	15.0	3.85	0.00	0.00	621000.
6	Cu Roug Co.	0.82	2.60	4.56	7.32	11.0	13.7	17.3	18.6	13.8	10.4	0.00	0.00	0.00	298000.
15	Cu Cl 1 Fd.	0.09	0.87	3.53	6.33	9.21	13.5	15.8	21.1	20.1	7.06	3.20	0.00	0.00	604000.
10	Cu Cl 1 Co.	0.75	2.81	5.92	9.60	13.6	15.2	18.5	14.4	11.7	7.48	0.00	0.00	0.00	486000.
12	Cu Cl 2 Co.	1.55	4.35	7.68	12.0	16.9	18.6	15.4	10.7	4.77	0.40	0.00	0.01	7.46	636000.
21	In Roug Fd.	15.9	84.1	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	32.
14	In Roug Co.	0.83	4.58	10.2	11.5	11.3	10.5	9.32	14.9	0.00	0.00	26.8	0.00	0.00	17400.
16	In Roug Co.	1.43	7.72	15.0	8.79	7.78	59.2	0.00	0.00	0.00	0.00	0.00	0.00	0.00	859.
18	In Roug Co.	0.35	1.42	7.18	7.99	14.1	5.17	8.33	15.9	39.6	0.00	0.00	0.00	0.00	3970.
20	In Roug Co.	2.36	6.58	13.0	26.6	12.7	0.00	0.00	38.8	0.00	0.00	0.00	0.00	0.00	1580.
22	In Roug Co.	0.00	0.45	2.97	7.71	7.27	25.6	11.0	11.7	0.00	33.3	0.00	0.00	0.00	6470.
24	In Cl 1 Co.	0.11	1.47	4.40	12.7	12.3	22.8	0.00	29.8	20.4	0.00	0.00	0.00	0.00	7600.
26	In Cl 2 Co.	0.00	31.7	68.3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	27.
3	Cu Roug Tl.	0.00	2.23	8.83	5.70	7.28	16.0	59.9	0.00	0.00	0.00	0.00	0.00	0.00	2190.
7	Cu Scav Tl.	0.62	4.01	9.02	24.3	14.8	17.7	13.0	16.5	0.00	0.00	0.00	0.00	0.00	4970.
9	Cu Scav Tl.	1.61	19.8	16.1	14.9	47.6	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	435.
11	Cu Cl 1 Tl.	0.23	1.56	5.05	7.95	12.4	12.9	12.2	20.2	14.2	7.99	5.39	0.00	0.00	96200.
25	Cu Cl 1 Tl.	0.04	0.16	2.61	4.48	8.67	11.0	16.5	8.96	20.8	20.8	5.93	0.00	0.00	77600.
13	Cu Cl 2 Tl.	0.41	4.26	10.3	15.8	17.9	16.1	14.4	9.42	3.64	4.33	3.36	0.00	0.00	136000.
23	In Cl 1 Tl.	0.00	6.13	18.7	28.0	17.9	0.00	29.3	0.00	0.00	0.00	0.00	0.00	0.00	962.
19	In Cl 2 Tl.	0.00	2.30	6.54	11.2	15.6	17.2	0.00	23.0	24.3	0.00	0.00	0.00	0.00	5170.
17	In Roug Tl.	0.00	10.3	19.8	0.00	69.8	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	101.

TABLE 2

THE GRAIN SIZE DISTRIBUTION OF FREE SPHALERITE IN THE 25 MILL PRODUCTS.

	Sample.	Classes (um). -3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	Area, (um^2)
1	Float Feed.	0.05	1.03	9.73	18.5	20.9	20.0	10.4	6.05	13.3	0.00	0.00	0.00	0.00	20900.
8	Cu Scav Co.	0.34	2.00	4.96	8.61	11.8	16.2	16.4	27.5	5.83	6.38	0.00	0.00	0.00	33400.
2	Cu Roug Co.	0.00	0.02	0.46	3.82	10.5	15.5	11.1	19.3	13.0	19.9	6.41	0.00	0.00	67500.
4	Cu Roug Co.	0.03	0.64	3.95	9.33	15.8	20.0	16.0	18.2	9.00	6.99	0.00	0.00	0.00	64000.
6	Cu Roug Co.	0.56	3.42	9.04	15.8	22.4	25.2	17.4	2.38	3.81	0.00	0.00	0.00	0.00	33100.
15	Cu Cl 1 Fd.	0.01	0.86	6.58	11.8	16.4	22.3	18.6	12.2	11.3	0.00	0.00	0.00	0.00	76100.
10	Cu Cl 1 Co.	1.30	4.25	9.42	13.4	20.8	19.1	14.4	17.3	0.00	0.00	0.00	0.00	0.00	51200.
12	Cu Cl 2 Co.	1.13	5.01	11.1	16.0	19.6	19.5	19.6	8.18	0.00	0.00	0.00	0.00	0.00	41500.
21	Zn Roug Fd.	1.71	4.32	7.68	13.2	17.5	16.0	9.96	4.40	7.61	17.7	0.00	0.00	0.00	14700.
14	Zn Roug Co.	0.01	0.37	2.41	5.67	9.25	13.3	17.1	19.5	20.0	12.3	0.00	0.00	0.00	460000.
16	Zn Roug Co.	0.38	2.29	5.63	8.91	13.1	18.9	21.0	18.6	6.00	5.41	0.00	0.00	0.00	163000.
18	Zn Roug Co.	1.06	2.64	4.90	7.15	9.86	13.1	17.4	21.7	10.7	9.75	1.82	0.00	0.00	243000.
20	Zn Roug Co.	1.84	5.03	9.34	13.3	17.2	18.5	16.6	11.3	3.60	1.72	1.51	0.00	0.00	332000.
22	Zn Roug Co.	0.08	0.72	2.28	4.25	6.25	9.88	14.1	20.0	18.1	13.9	10.4	0.00	0.00	852000.
24	Zn Cl 1 Co.	0.11	1.30	4.06	6.91	10.3	13.7	15.7	18.2	17.0	7.26	5.59	0.00	0.00	742000.
26	Zn Cl 2 Co.	3.08	3.85	6.00	8.19	12.5	14.2	17.4	15.6	13.4	5.78	0.00	0.00	0.00	359000.
3	Cu Roug Tl.	0.00	0.00	0.00	2.49	6.62	18.6	12.5	13.9	13.3	32.6	0.00	0.00	0.00	27500.
7	Cu Scav Tl.	0.00	0.12	0.99	3.96	6.91	17.3	16.9	11.7	30.9	3.42	7.85	0.00	0.00	63900.
9	Cu Scav Tl.	0.00	0.32	2.24	6.17	10.0	17.9	22.0	9.32	25.4	6.66	0.00	0.00	0.00	32800.
11	Cu Cl 1 Tl.	0.00	0.57	4.98	12.2	15.8	19.6	24.7	12.4	9.87	0.00	0.00	0.00	0.00	59900.
25	Cu Cl 1 Tl.	0.14	1.26	3.24	6.22	9.15	13.6	19.2	23.3	11.9	12.0	0.00	0.00	0.00	105000.
13	Cu Cl 2 Tl.	0.81	4.87	11.2	16.2	20.3	18.4	14.4	9.79	4.08	0.00	0.00	0.00	0.00	72100.
23	Zn Cl 1 Tl.	0.96	6.84	18.4	17.2	11.8	13.0	13.2	18.6	0.00	0.00	0.00	0.00	0.00	4380.
19	Zn Cl 2 Tl.	0.46	4.95	7.29	10.6	10.7	12.2	8.56	29.3	16.0	0.00	0.00	0.00	0.00	13000.
17	Zn Roug Tl.	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.

TABLE 3

THE GRAIN SIZE DISTRIBUTION OF FREE GALENA IN THE 25 MILL PRODUCTS.

Sample.	Classes (um)												Area, (um^2)
	-1.15	-1.62	-2.30	-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0	
1 Float Feed (Run 1).	0.00	1.27	2.62	5.19	4.64	23.7	0.00	62.6	0.00	0.00	0.00	0.00	0.00	306.
1 Float Feed (Run 2).	0.00	0.00	1.37	7.83	14.7	11.1	65.0	0.00	0.00	0.00	0.00	0.00	0.00	156.
8 Cu Scav Co.	0.00	0.00	0.00	0.00	3.66	17.3	11.1	46.1	22.7	0.00	0.00	0.00	0.00	1170.
2 Cu Roug Co (Maj M).	-----	-----	-----	4.57	5.21	6.45	6.56	17.9	17.7	11.0	30.7	0.00	0.00	3690.
4 Cu Roug Co.	0.00	3.55	8.58	13.7	24.7	11.6	38.5	0.00	0.00	0.00	0.00	0.00	0.00	193.
6 Cu Roug Co.	0.00	0.92	3.13	13.9	25.0	17.5	40.5	0.00	0.00	0.00	0.00	0.00	0.00	805.
15 Cu Cl 1 Fd.	0.00	0.00	1.08	6.28	16.2	9.99	12.5	0.00	56.0	0.00	0.00	0.00	0.00	279.
10 Cu Cl 1 Co.	0.00	2.22	6.47	22.2	26.0	9.06	13.8	20.3	0.00	0.00	0.00	0.00	0.00	647.
12 Cu Cl 2 Co (Run 1).	0.00	0.28	1.56	1.85	18.7	6.36	32.6	18.8	37.8	0.00	0.00	0.00	0.00	697.
12 Cu Cl 2 Co (Run 2).	0.00	0.00	3.06	4.29	59.7	0.00	33.0	0.00	0.00	0.00	0.00	0.00	0.00	126.
21 Zn Roug Fd.	0.00	0.00	0.00	0.27	5.63	21.0	8.57	24.1	21.1	19.3	0.00	0.00	0.00	1600.
14 Zn Roug Co (Run 1).	0.00	0.36	0.80	5.87	9.83	14.1	3.97	21.4	6.78	0.00	37.2	0.00	0.00	2070.
14 Zn Roug Co (Run 2).	0.00	2.71	7.62	17.0	2.68	16.9	0.00	53.2	0.00	0.00	0.00	0.00	0.00	344.
16 Zn Roug Co (Maj M).	-----	-----	-----	1.66	5.76	6.10	8.32	5.79	21.4	14.7	0.00	36.2	0.00	3380.
18 Zn Roug Co.	0.00	0.00	0.02	0.00	0.38	1.62	7.15	8.29	11.0	16.2	23.4	24.3	7.71	36300.
20 Zn Roug Co (Maj M).	-----	-----	-----	10.9	9.44	12.2	10.4	19.7	6.47	0.00	31.0	0.00	0.00	2420.
22 Zn Roug Co.	0.00	0.72	3.42	7.82	10.8	15.0	12.6	18.9	11.8	6.63	12.3	0.00	0.00	4520.
24 Zn Cl 1 Co.	0.00	0.17	1.30	1.66	3.03	6.8	9.65	13.0	33.5	30.7	0.00	0.00	0.00	5850.
26 Zn Cl 2 Co.	0.00	2.48	6.22	10.0	16.6	5.34	23.2	17.6	18.6	0.00	0.00	0.00	0.00	1360
3 Cu Roug Tl.	0.00	0.00	0.00	0.00	1.47	5.21	6.49	5.35	8.98	20.9	51.6	0.00	0.00	3930.
7 Cu Scav Tl.	0.85	4.38	3.08	6.98	7.97	26.6	0.00	52.1	0.00	0.00	0.00	0.00	0.00	197.
9 Cu Scav Tl.	0.00	0.53	5.53	13.6	16.9	19.7	26.5	17.4	0.00	0.00	0.00	0.00	0.00	693.
11 Cu Cl 1 Tl (Run 1).	0.00	0.00	0.00	0.00	0.48	2.08	9.97	24.1	18.0	14.5	5.30	0.00	25.5	11500.
11 Cu Cl 1 Tl (Run 2).	0.00	0.00	0.00	0.00	13.6	86.4	0.00	0.00	0.00	0.00	0.00	0.00	0.00	226.
25 Cu Cl 1 Tl (Maj M).	-----	-----	-----	3.36	8.40	0.00	0.00	0.00	88.2	0.00	0.00	0.00	0.00	238.
13 Cu Cl 2 Tl.	0.00	0.00	0.00	0.00	1.32	9.41	15.8	19.4	13.4	16.6	22.9	0.00	0.00	5980.
23 Zn Cl 1 Tl.	0.00	0.00	0.00	0.00	0.05	0.29	3.55	7.33	5.34	14.2	39.8	12.0	28.5	31300.
19 Zn Cl 2 Tl.	0.00	0.23	0.82	1.32	2.80	4.22	7.98	14.5	26.6	33.7	9.36	8.44	0.00	25300.
17 Zn Roug Tl (Run 1).	0.00	0.00	0.00	0.00	0.00	5.71	47.2	6.59	13.1	27.4	0.00	0.00	0.00	1020.
17 Zn Roug Tl (Run 2).	0.00	0.00	0.00	0.00	1.66	9.11	28.9	15.1	45.2	0.00	0.00	0.00	0.00	845.

Note : (Maj M) denotes data generated by the Major Mineral Routine.

TABLE 4

THE GRAIN SIZE DISTRIBUTION OF UNLIBERATED CHALCOPYRITE IN THE 25 MILL PRODUCTS.

	Sample.	Classes (um).												Area, (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	
1	Float Feed.	0.17	1.35	7.79	9.89	16.8	18.9	27.3	12.4	0.00	0.00	0.00	0.00	0.00	25000.
8	Cu Scav Co.	0.26	0.69	1.15	2.43	4.30	5.30	9.18	8.01	18.5	19.9	15.9	14.4	0.00	295000.
2	Cu Roug Co.	0.07	0.17	0.69	2.48	4.02	6.83	9.36	17.5	20.6	31.7	6.70	0.00	0.00	151000.
4	Cu Roug Co.	0.06	0.35	1.09	1.67	2.72	5.12	8.79	13.1	18.1	32.3	13.9	2.74	0.00	420000.
6	Cu Roug Co.	0.34	1.04	1.64	3.43	6.61	7.82	13.3	17.2	28.4	16.9	3.25	0.00	0.00	209000.
15	Cu Cl 1 Fd.	0.12	0.67	1.61	2.72	3.68	7.74	10.4	16.9	27.8	14.4	13.9	0.00	0.00	329000.
10	Cu Cl 1 Co.	0.33	0.94	1.83	3.62	5.46	8.63	13.6	19.3	18.2	13.9	0.00	14.2	0.00	253000.
12	Cu Cl 2 Co.	0.81	1.87	3.38	6.20	10.5	16.9	22.0	23.1	11.0	1.17	3.03	0.00	0.00	243000.
21	In Roug Fd.	47.9	0.00	8.07	13.4	13.4	17.3	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1680.
14	In Roug Co.	0.81	3.49	8.77	12.4	16.1	12.3	14.6	13.4	12.3	7.77	0.00	0.00	0.00	77500.
16	In Roug Co.	1.18	4.28	3.91	10.7	8.10	0.00	0.00	28.5	0.00	43.3	0.00	0.00	0.00	9640.
18	In Roug Co.	0.58	2.73	3.76	7.82	10.4	15.2	12.8	10.7	15.6	20.3	0.00	0.00	0.00	30700.
20	In Roug Co.	1.78	7.65	8.04	11.9	13.8	8.94	15.4	32.5	0.00	0.00	0.00	0.00	0.00	10500.
22	In Roug Co.	0.03	1.97	5.75	10.9	13.6	22.2	23.7	21.8	0.00	0.00	0.00	0.00	0.00	29200.
24	In Cl 1 Co.	0.04	1.60	5.37	9.10	10.9	13.7	17.2	16.7	15.6	9.75	0.00	0.00	0.00	47400.
26	In Cl 2 Co.	0.89	3.43	10.0	10.8	16.7	28.4	3.92	25.9	0.00	0.00	0.00	0.00	0.00	11200.
3	Cu Roug Tl.	7.63	0.63	6.36	7.04	2.40	12.6	21.9	0.00	41.4	0.00	0.00	0.00	0.00	6380.
7	Cu Scav Tl.	11.9	2.23	4.35	6.01	16.2	16.3	12.7	8.80	21.5	0.00	0.00	0.00	0.00	8310.
9	Cu Scav Tl.	1.30	1.06	1.84	2.82	6.85	1.84	3.07	21.3	0.00	59.9	0.00	0.00	0.00	12700.
11	Cu Cl 1 Tl.	0.27	1.56	3.83	7.17	11.5	12.2	16.3	13.6	16.2	17.4	0.00	0.00	0.00	140000.
25	Cu Cl 1 Tl.	0.01	0.42	2.07	5.37	9.71	12.7	12.6	17.3	17.4	22.5	0.00	0.00	0.00	88400.
13	Cu Cl 2 Tl.	0.40	2.14	4.81	8.59	14.0	13.9	14.1	18.2	19.4	4.44	0.00	0.00	0.00	62700.
23	In Cl 1 Tl.	2.69	1.18	7.18	11.4	13.3	25.2	14.4	0.00	24.7	0.00	0.00	0.00	0.00	7620.
19	In Cl 2 Tl.	0.00	0.61	3.21	4.36	11.2	13.8	9.91	11.2	11.8	4.87	29.0	0.00	0.00	66100.
17	In Roug Tl.	1.50	1.89	0.98	9.96	9.19	6.98	26.5	43.0	0.00	0.00	0.00	0.00	0.00	2480.

TABLE 5

THE GRAIN SIZE DISTRIBUTION OF UNLIBERATED SPHALERITE IN THE 25 MILL PRODUCTS.

	Sample.	Classes (um).												Area. (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	
1	Float Feed.	0.00	0.05	0.79	4.47	7.38	17.9	11.6	25.6	9.96	22.2	0.00	0.00	0.00	31700.
8	Cu Scav Co.	0.05	0.27	0.70	1.91	2.38	6.01	10.6	19.7	33.7	20.1	4.54	0.00	0.00	199000.
2	Cu Roug Co.	0.00	0.00	0.26	2.70	7.00	8.40	19.1	12.6	30.8	19.2	0.00	0.00	0.00	93900.
4	Cu Roug Co.	0.01	0.21	1.06	2.61	4.81	9.33	20.6	22.1	21.3	11.9	5.96	0.00	0.00	117000.
6	Cu Roug Co.	0.29	1.39	2.41	4.70	11.4	13.8	22.1	12.4	19.6	11.9	0.00	0.00	0.00	50300.
15	Cu Cl 1 Fd.	0.01	0.33	0.95	2.83	5.56	11.2	12.2	20.8	24.2	21.9	0.00	0.00	0.00	148000.
10	Cu Cl 1 Co.	0.39	1.33	3.74	6.49	8.37	20.6	16.1	14.5	23.1	5.55	0.00	0.00	0.00	62500.
12	Cu Cl 2 Co.	0.70	3.24	7.52	13.6	16.8	19.6	27.8	10.9	0.00	0.00	0.00	0.00	0.00	40900.
21	In Roug Fd.	0.89	1.34	1.29	4.50	10.6	6.86	9.54	8.54	56.4	0.00	0.00	0.00	0.00	11000.
14	In Roug Co.	0.00	0.08	0.36	0.95	2.28	4.37	10.6	13.8	25.6	25.7	12.2	4.36	0.00	724000.
16	In Roug Co.	0.11	0.55	2.15	4.22	6.43	11.8	21.0	22.7	26.7	4.63	0.00	0.00	0.00	62200.
18	In Roug Co.	0.24	0.49	0.85	1.78	3.82	7.40	13.9	21.6	24.2	19.0	6.74	0.00	0.00	262000.
20	In Roug Co.	0.55	1.46	2.56	5.48	9.51	10.2	27.1	22.7	20.4	0.00	0.00	0.00	0.00	116000.
22	In Roug Co.	0.03	0.16	0.48	0.81	2.15	4.63	8.13	18.7	26.5	22.6	13.4	2.45	0.00	498000.
24	In Cl 1 Co.	0.03	0.24	0.72	1.36	2.64	6.29	10.8	14.6	22.2	25.4	13.1	2.69	0.00	457000.
26	In Cl 2 Co.	0.29	0.44	0.80	2.02	3.05	7.97	11.2	22.3	35.6	16.3	0.00	0.00	0.00	209000.
3	Cu Roug Tl.	0.05	0.00	0.08	1.24	3.61	8.93	7.32	19.3	43.6	15.9	0.00	0.00	0.00	40200.
7	Cu Scav Tl.	0.14	0.23	0.38	1.64	3.73	8.24	8.82	13.8	40.6	22.5	0.00	0.00	0.00	44500.
9	Cu Scav Tl.	0.17	0.21	0.58	0.83	2.11	3.41	1.99	18.6	19.2	11.9	0.00	41.0	0.00	48400.
11	Cu Cl 1 Tl.	0.01	0.11	0.87	2.45	5.62	11.9	19.7	20.3	24.3	14.7	0.00	0.00	0.00	180000.
25	Cu Cl 1 Tl.	0.02	0.27	0.58	1.84	3.90	4.27	12.7	17.2	30.1	19.0	11.1	0.00	0.00	113000.
13	Cu Cl 2 Tl.	0.32	0.97	3.70	7.15	10.7	19.7	21.0	27.0	9.54	0.00	0.00	0.00	0.00	37500.
23	In Cl 1 Tl.	0.45	3.34	5.06	11.4	16.3	11.2	27.0	0.00	25.2	0.00	0.00	0.00	0.00	15800.
19	In Cl 2 Tl.	0.23	1.62	4.13	8.38	12.2	16.6	17.8	21.4	14.6	3.04	0.00	0.00	0.00	93100.
17	In Roug Tl.	0.00	1.39	2.44	7.23	24.9	30.0	0.00	44.1	0.00	0.00	0.00	0.00	0.00	2420.

TABLE 6

THE GRAIN SIZE DISTRIBUTION OF UNLIBERATED GALENA IN THE 25 MILL PRODUCTS.

Sample.	Classes (um)												Area (um²)
	-1.15	-1.62	-2.30	-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	+52.0	
1 Float Feed (Run 1).	0.00	0.06	0.53	0.76	2.41	4.74	20.1	28.5	31.4	11.5	0.00	0.00	0.00	3010.
1 Float Feed (Run 2).	0.00	1.72	4.19	10.1	6.87	15.4	17.5	5.11	10.2	29.0	0.00	0.00	0.00	2130.
8 Cu Scav Co.	0.00	0.00	0.00	0.00	1.37	3.19	6.05	13.4	24.5	16.9	14.4	20.2	0.00	12400.
2 Cu Roug Co (Maj M).	-----	-----	-----	0.00	46.0	54.0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1890.
4 Cu Roug Co.	0.00	0.80	1.28	4.47	14.8	11.7	18.1	22.1	0.00	27.1	0.00	0.00	0.00	1710.
6 Cu Roug Co.	0.00	0.26	5.38	10.7	23.7	16.3	17.1	15.1	11.3	0.00	0.00	0.00	0.00	1750.
15 Cu Cl 1 Fd.	0.00	0.53	3.48	9.87	22.3	14.6	18.8	30.5	0.00	0.00	0.00	0.00	0.00	712.
10 Cu Cl 1 Co.	0.00	1.09	4.63	6.96	16.1	18.6	11.5	33.7	7.65	0.00	0.00	0.00	0.00	2680.
12 Cu Cl 2 Co (Run 1).	0.00	0.19	1.46	4.08	14.5	16.1	36.9	4.76	0.00	22.0	0.00	0.00	0.00	2160.
12 Cu Cl 2 Co (Run 2).	0.00	0.00	2.28	7.57	25.8	15.2	30.8	19.3	0.00	0.00	0.00	0.00	0.00	913.
21 In Roug Fd.	0.00	0.00	0.00	0.23	0.96	3.47	5.78	9.39	12.2	10.6	8.56	5.52	41.7	27700.
14 In Roug Co (Run 1).	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.
14 In Roug Co (Run 2).	0.00	0.56	3.99	8.18	7.66	17.2	18.7	17.8	0.00	25.6	0.00	0.00	0.00	1370.
16 In Roug Co (Maj M).	-----	-----	-----	0.65	18.6	40.3	40.5	0.00	0.00	0.00	0.00	0.00	0.00	5790.
18 In Roug Co.	0.00	0.00	0.00	0.04	0.41	2.21	4.76	9.11	11.5	11.2	17.7	10.2	32.9	101000.
20 In Roug Co (Maj M).	-----	-----	-----	2.61	38.4	16.1	42.9	0.00	0.00	0.00	0.00	0.00	0.00	4010.
22 In Roug Co.	0.02	1.13	2.92	6.57	8.79	13.5	16.3	23.8	10.3	16.8	0.00	0.00	0.00	7990.
24 In Cl 1 Co.	0.01	5.62	1.05	2.50	5.67	4.83	6.81	9.59	20.8	3.02	36.0	11.5	0.00	12300
26 In Cl 2 Co.	0.03	0.47	1.85	4.29	5.82	5.47	12.0	8.33	11.8	14.2	0.00	35.8	0.00	9530
3 Cu Roug Tl.	0.00	0.00	0.00	0.00	0.43	2.50	4.74	9.78	9.74	7.52	14.3	13.7	37.3	58485.
7 Cu Scav Tl.	0.05	5.30	1.73	1.32	3.73	5.76	5.96	7.75	17.0	13.3	43.1	0.00	0.00	4080.
9 Cu Scav Tl.	0.00	0.27	1.30	4.66	8.71	10.3	11.7	3.44	8.99	17.5	33.1	0.00	0.00	8420.
11 Cu Cl 1 Tl (Run 1).	0.00	0.00	0.00	0.00	0.27	1.90	4.46	10.4	17.0	22.4	12.9	20.6	10.2	39000.
11 Cu Cl 1 Tl (Run 2).	0.00	0.00	0.00	0.00	21.3	13.7	0.00	65.0	0.00	0.00	0.00	0.00	0.00	7370.
25 Cu Cl 1 Tl (Maj M).	-----	-----	-----	0.05	9.45	32.1	33.2	25.2	0.00	0.00	0.00	0.00	0.00	17400.
13 Cu Cl 2 Tl.	0.00	0.00	0.00	0.03	0.77	3.19	6.52	9.32	11.6	16.2	11.1	26.5	15.8	96200.
23 In Cl 1 Tl.	0.00	0.00	0.00	0.00	0.00	0.15	1.86	5.09	9.45	10.4	13.8	25.8	37.4	58500.
19 In Cl 2 Tl.	0.00	0.10	0.25	0.87	1.82	3.10	3.58	9.36	18.0	29.6	26.8	7.45	0.00	41900.
17 In Roug Tl (Run 1).	0.00	0.00	0.00	0.00	0.00	1.83	0.00	4.78	8.60	12.9	0.00	0.00	72.2	10300.
17 In Roug Tl (Run 2).	0.00	0.00	0.00	0.01	0.11	0.09	0.38	2.04	3.36	5.94	6.21	0.00	81.9	94000.

Note : (Maj M) denotes data generated by the Major Mineral Routine.

TABLE 7

THE GRAIN SIZE DISTRIBUTION OF SUB-FREE CHALCOPYRITE IN THE 25 MILL PRODUCTS.

	Sample.	Classes (um).												Area. (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	
1	Float Feed.	0.00	0.45	2.90	5.00	6.29	10.5	12.4	18.5	12.4	0.00	0.00	0.00	0.00	17040.
8	Cu Scav Co.	0.10	0.01	0.32	0.73	1.45	1.29	3.72	5.69	14.7	19.9	13.7	14.4	0.00	225720.
2	Cu Roug Co.	0.00	0.00	0.02	0.08	0.87	3.15	4.68	13.0	12.4	26.4	6.78	0.00	0.00	101700.
4	Cu Roug Co.	0.02	0.15	0.45	0.83	1.54	3.99	6.69	12.0	17.6	32.3	13.9	2.74	0.00	387420.
6	Cu Roug Co.	0.16	0.49	1.00	2.42	4.64	6.29	11.8	15.6	27.7	16.9	3.25	0.00	0.00	189100.
15	Cu Cl 1 Fd.	0.02	0.20	0.54	1.00	1.85	4.83	12.6	13.1	26.3	14.4	13.9	0.00	0.00	278470.
10	Cu Cl 1 Co.	0.13	0.44	0.59	2.39	3.42	6.39	11.7	18.5	17.5	13.9	0.00	14.2	0.00	226700.
12	Cu Cl 2 Co.	0.46	1.27	2.43	5.43	9.39	15.6	20.9	22.7	11.3	1.17	3.03	0.00	0.00	227060.
21	In Roug Fd.	0.24	0.00	1.46	10.7	7.16	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	330.
14	In Roug Co.	0.03	0.27	0.25	0.53	1.34	1.98	5.64	2.43	6.53	3.77	0.00	0.00	0.00	22800.
16	In Roug Co.	0.18	0.13	0.00	0.57	0.00	0.00	0.00	0.00	0.00	43.3	0.00	0.00	0.00	4260.
18	In Roug Co.	0.05	0.39	0.47	1.05	3.08	1.79	1.53	5.73	10.7	20.3	0.00	0.00	0.00	14153.
20	In Roug Co.	0.00	0.68	1.81	2.13	3.13	4.00	15.4	17.4	0.00	0.00	0.00	0.00	0.00	4675.
22	In Roug Co.	0.00	0.13	0.36	2.03	2.96	1.69	9.21	2.34	0.00	0.00	0.00	0.00	0.00	5457.
24	In Cl 1 Co.	0.00	0.06	0.86	1.40	1.91	4.81	4.30	6.26	9.37	0.00	0.00	0.00	0.00	13531.
26	In Cl 2 Co.	0.00	0.00	0.58	1.22	1.91	9.39	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1465.
3	Cu Roug Tl.	0.00	0.00	0.00	0.00	0.00	4.76	0.00	0.00	0.00	0.00	0.00	0.00	0.00	304.
7	Cu Scav Tl.	0.00	0.95	1.04	3.08	1.07	3.34	7.38	0.00	21.5	0.00	0.00	0.00	0.00	3106.
9	Cu Scav Tl.	0.04	0.09	0.82	0.86	1.50	0.00	0.00	8.85	0.00	36.3	0.00	0.00	0.00	6150.
11	Cu Cl 1 Tl.	0.02	0.17	0.52	1.33	2.96	2.73	7.27	6.18	8.27	17.4	0.00	0.00	0.00	65670.
25	Cu Cl 1 Tl.	0.00	0.15	0.61	1.54	3.96	3.36	5.40	8.49	15.5	22.5	0.00	0.00	0.00	54280.
13	Cu Cl 2 Tl.	0.13	0.66	1.73	2.95	6.21	8.09	12.2	16.1	16.9	4.44	0.00	0.00	0.00	43580.
23	In Cl 1 Tl.	0.00	0.47	1.65	5.51	1.44	17.7	7.36	0.00	0.00	0.00	0.00	0.00	0.00	2610.
19	In Cl 2 Tl.	0.00	0.14	0.61	1.21	4.72	2.74	4.05	6.84	8.56	4.87	29.0	0.00	0.00	61490.
17	In Roug Tl.	0.00	0.00	0.00	0.00	0.00	0.00	0.00	43.0	0.00	0.00	0.00	0.00	0.00	1070.

NOTE : Data presented as areal percent of unliberated chalcopyrite occurring as sub-free (70-99.9%) chalcopyrite, occurring in each grain size class.

TABLE 8

THE GRAIN SIZE DISTRIBUTION OF SUB-FREE SPHALERITE IN THE 25 MILL PRODUCTS.

	Sample.	Classes (um).												Area. (um^2)
		-3.25	-4.60	-6.50	-9.19	-13.0	-18.4	-26.0	-36.8	-52.0	-73.5	-104.	-147.	-208.	
1	Float Feed.	0.00	0.00	0.69	2.70	4.56	14.1	9.85	18.8	9.96	22.2	0.00	0.00	0.00	26240.
8	Cu Scav Co.	0.01	0.02	0.14	0.68	1.21	3.33	7.55	16.0	30.9	20.1	4.54	0.00	0.00	167580.
2	Cu Roug Co.	0.00	0.00	0.04	0.72	2.94	3.64	14.5	6.48	21.6	13.6	0.00	0.00	0.00	59540.
4	Cu Roug Co.	0.00	0.02	0.33	1.27	2.77	5.96	14.4	15.3	20.0	11.9	5.96	0.00	0.00	90800.
6	Cu Roug Co.	0.00	0.26	0.74	3.15	7.09	9.43	16.3	6.31	19.6	6.02	0.00	0.00	0.00	34610.
15	Cu Cl 1 Fd.	0.00	0.01	0.14	1.09	3.41	7.74	9.58	16.5	20.1	0.00	0.00	0.00	0.00	106200.
10	Cu Cl 1 Co.	0.01	0.23	0.92	2.76	6.11	14.1	13.1	12.6	23.1	5.55	0.00	0.00	0.00	48970.
12	Cu Cl 2 Co.	0.15	0.63	3.84	9.79	10.4	15.6	25.7	12.1	0.00	0.00	0.00	0.00	0.00	28890.
21	Zn Roug Fd.	0.26	0.13	0.34	1.76	7.93	5.05	4.66	8.54	56.4	0.00	0.00	0.00	0.00	9321.
14	Zn Roug Co.	0.00	0.01	0.10	0.39	1.62	3.52	8.40	12.2	22.9	24.6	11.4	4.36	0.00	645900.
16	Zn Roug Co.	0.04	0.14	0.92	1.31	4.63	8.04	17.3	22.7	24.3	4.63	0.00	0.00	0.00	52320.
18	Zn Roug Co.	0.04	0.20	0.38	0.81	1.90	4.36	9.59	17.7	23.1	17.7	6.74	0.00	0.00	216500.
20	Zn Roug Co.	0.18	0.69	1.43	3.15	6.93	8.28	22.5	21.2	17.6	0.00	0.00	0.00	0.00	94740.
22	Zn Roug Co.	0.00	0.03	0.19	0.15	1.06	2.12	4.71	13.5	26.2	22.6	13.3	2.45	0.00	421400.
24	Zn Cl 1 Co.	0.00	0.05	0.23	0.58	0.93	3.37	7.57	12.5	19.4	23.1	13.1	2.69	0.00	381400.
26	Zn Cl 2 Co.	0.06	0.08	0.24	1.09	1.81	5.57	9.37	18.1	32.1	16.3	0.00	0.00	0.00	177590.
3	Cu Roug Tl.	0.00	0.00	0.08	0.53	2.52	6.00	6.42	17.1	39.1	15.9	0.00	0.00	0.00	35200.
7	Cu Scav Tl.	0.00	0.04	0.09	0.68	2.38	2.89	5.59	12.2	35.9	22.4	0.00	0.00	0.00	36641.
9	Cu Scav Tl.	0.00	0.03	0.12	0.28	0.54	1.15	1.27	13.3	16.3	5.77	0.00	41.0	0.00	38620.
11	Cu Cl 1 Tl.	0.00	0.03	0.09	1.00	3.42	8.43	15.8	17.8	23.4	14.7	0.00	0.00	0.00	152300.
25	Cu Cl 1 Tl.	0.00	0.04	0.05	0.48	1.41	1.97	9.45	12.4	28.9	16.5	11.1	0.00	0.00	93000.
13	Cu Cl 2 Tl.	0.08	0.16	1.26	2.70	7.12	12.8	18.0	25.2	9.54	0.00	0.00	0.00	0.00	28770.
23	Zn Cl 1 Tl.	0.04	0.08	0.44	0.00	0.00	0.00	5.81	0.00	9.22	0.00	0.00	0.00	0.00	2468.
19	Zn Cl 2 Tl.	0.00	0.04	0.06	0.09	0.77	1.28	0.37	2.04	1.60	0.00	0.00	0.00	0.00	5810.
17	Zn Roug Tl.	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.

NOTE : Data presented as areal percent of unliberated sphalerite occurring as sub-free (70-99.9%) sphalerite, occurring in each grain size class.

Appendix D

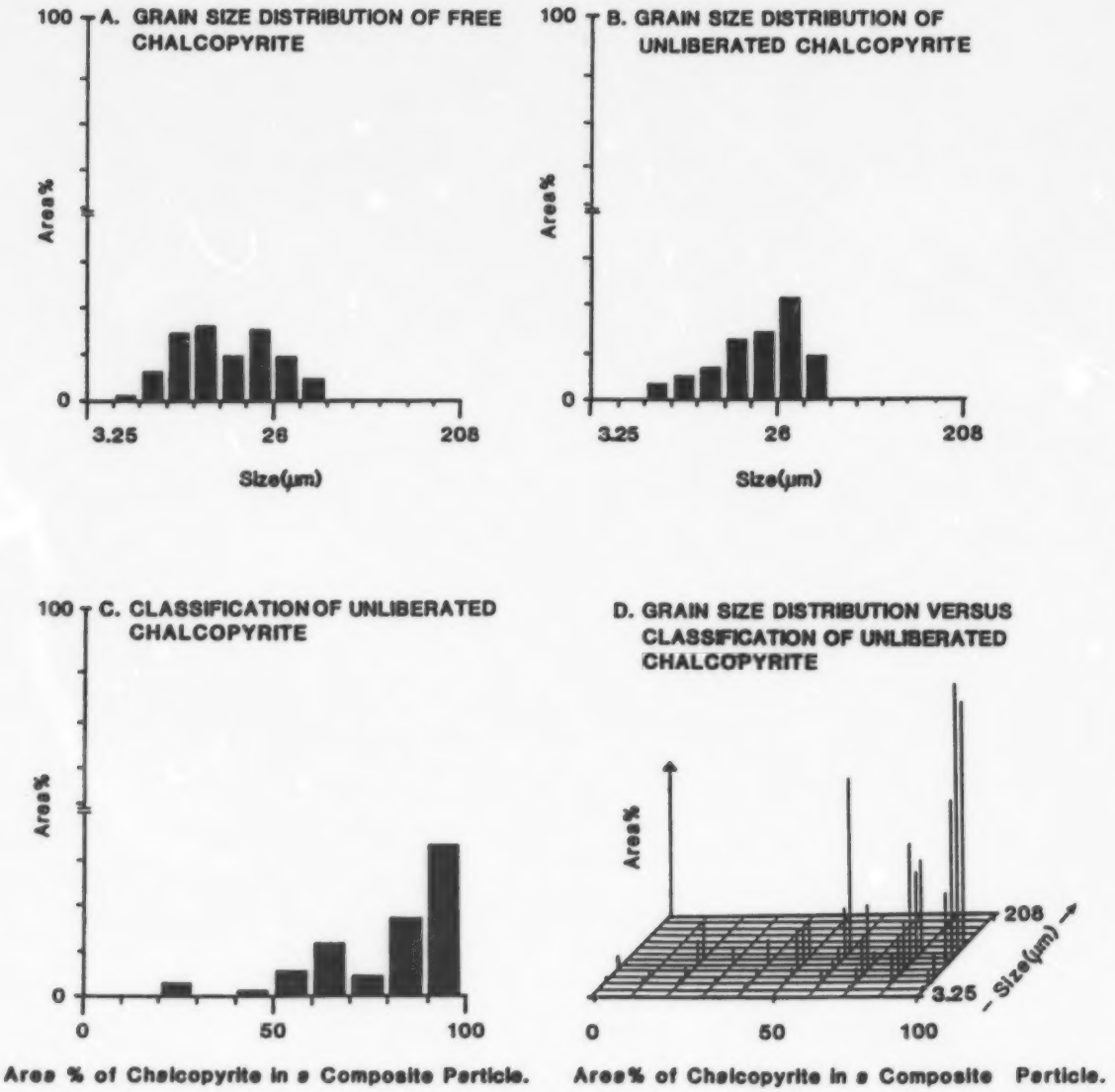

FIGURE 1
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE FLOAT FEED.

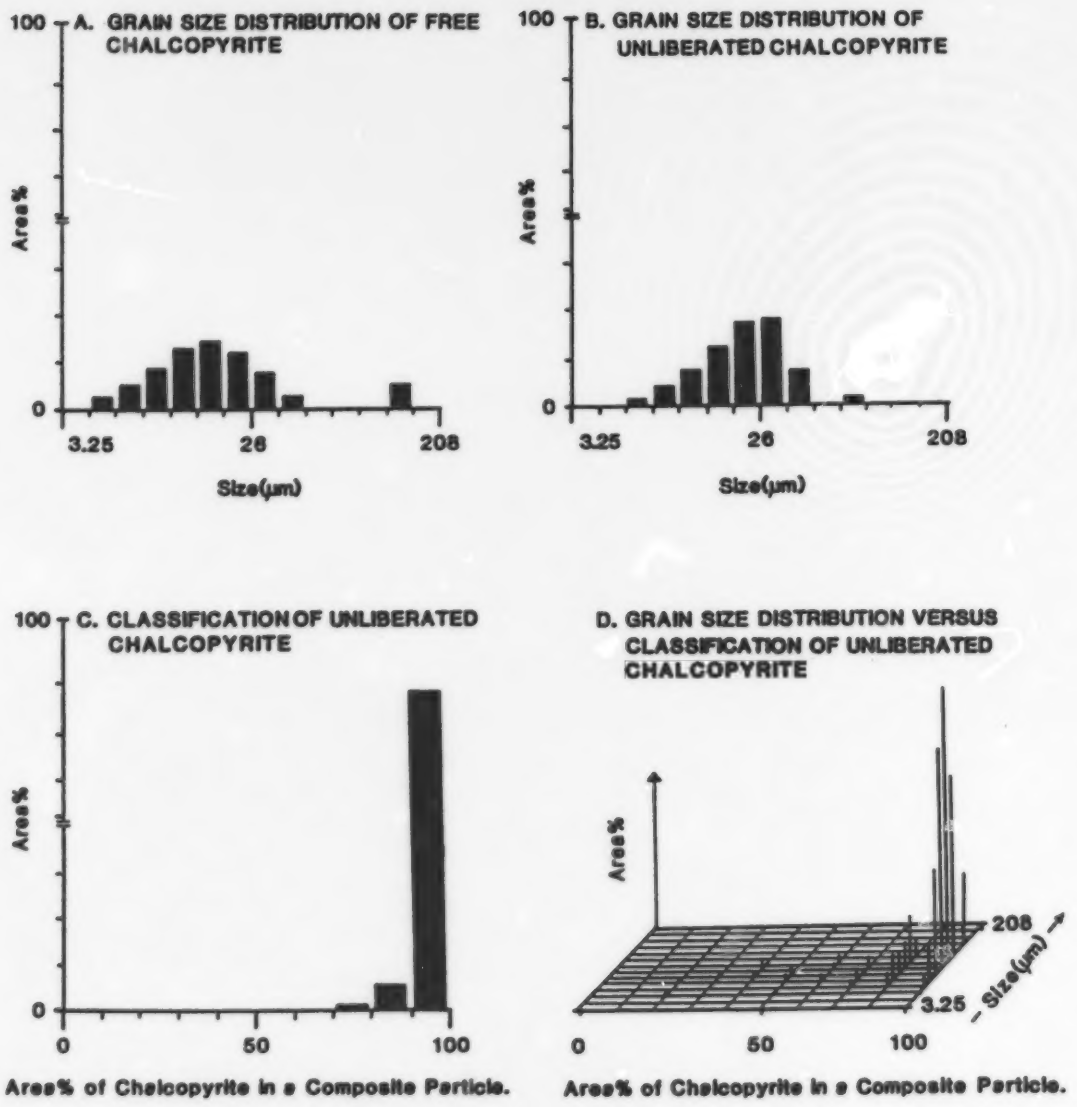

FIGURE 2.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE Cu-CONCENTRATE.

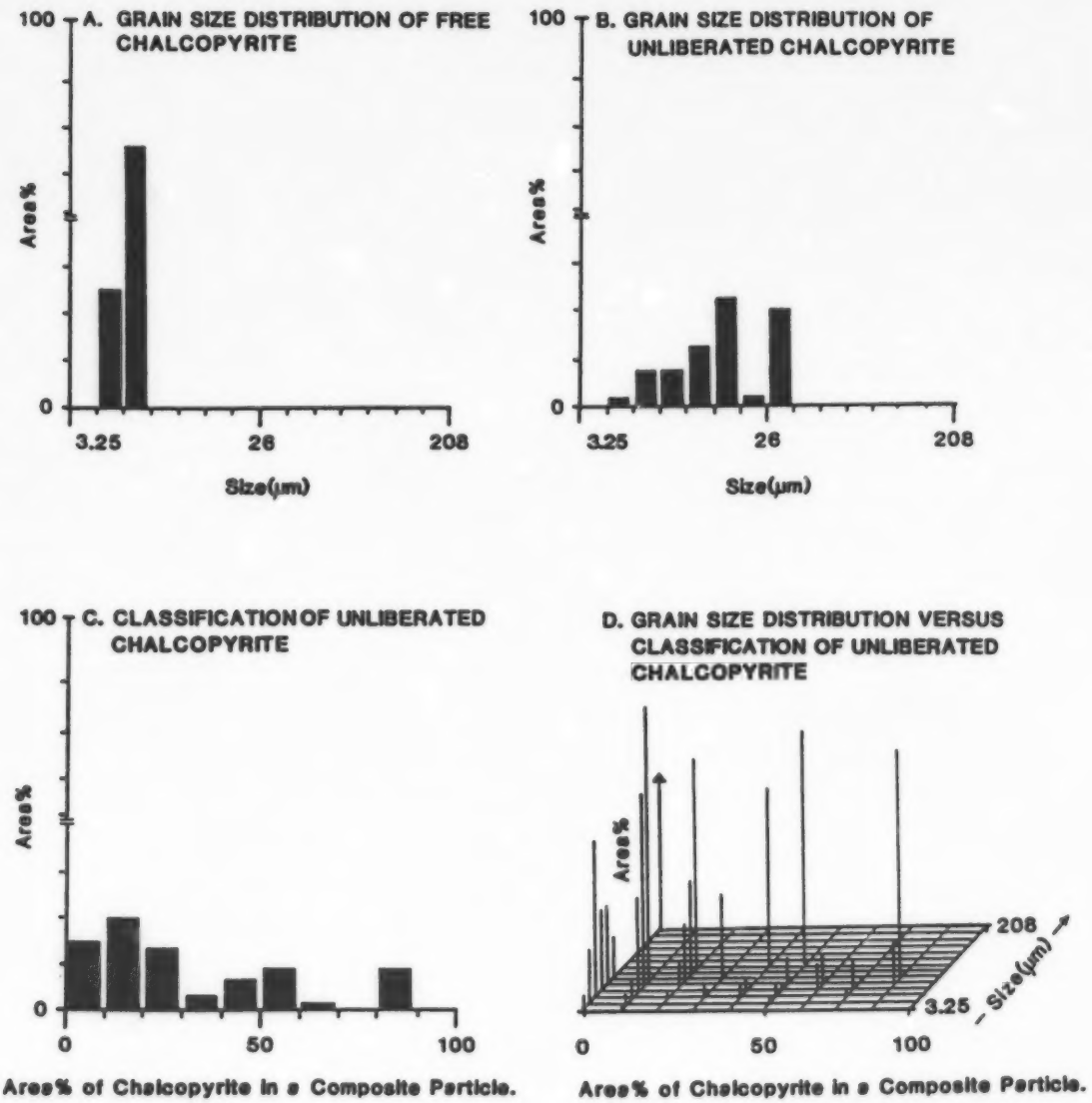

FIGURE 3.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE Zn-CONCENTRATE.

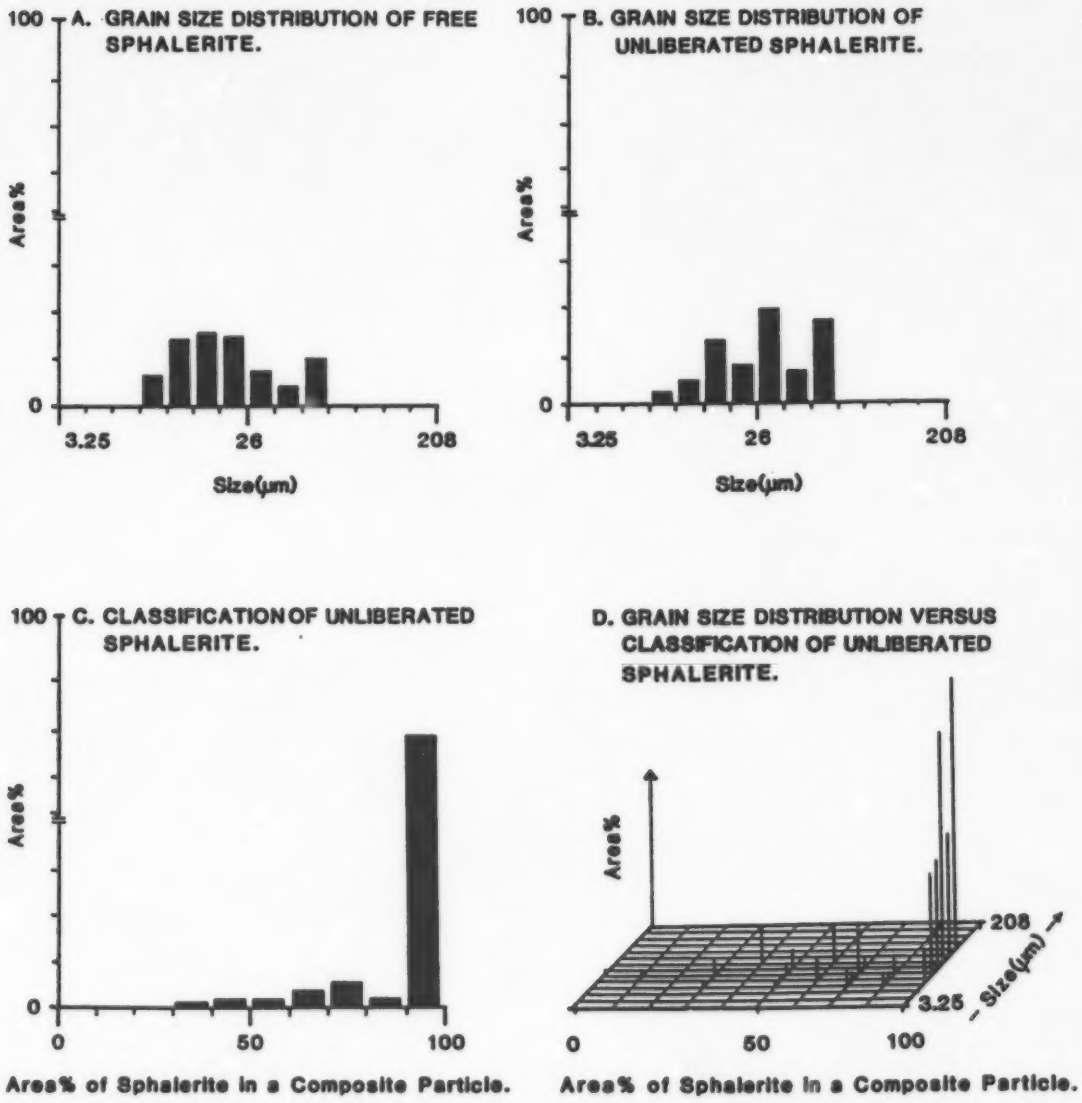

FIGURE 4.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR SPHALERITE IN THE FLOAT FEED.

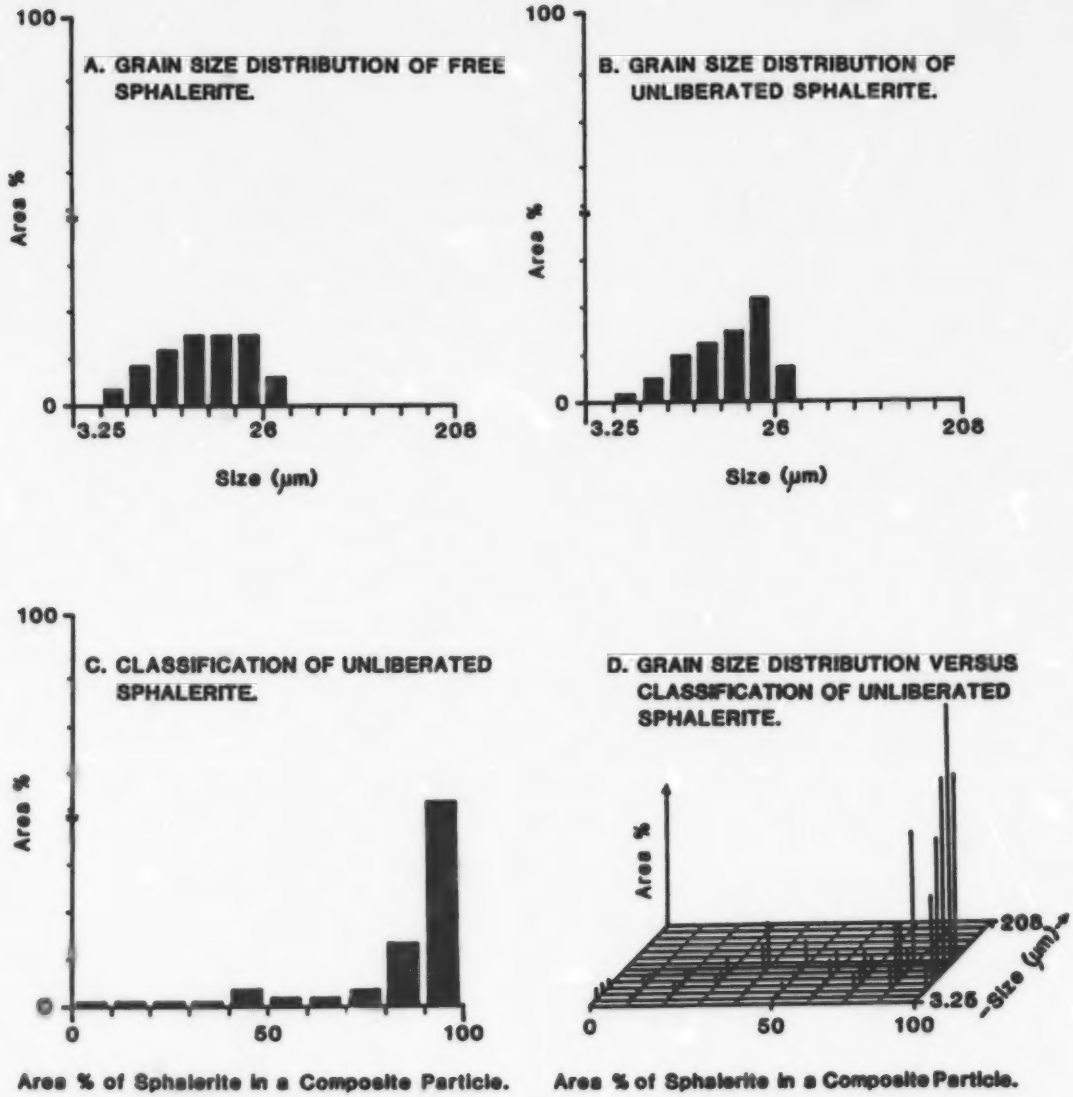

FIGURE 5.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR SPHALERITE IN THE Cu-CONCENTRATE.

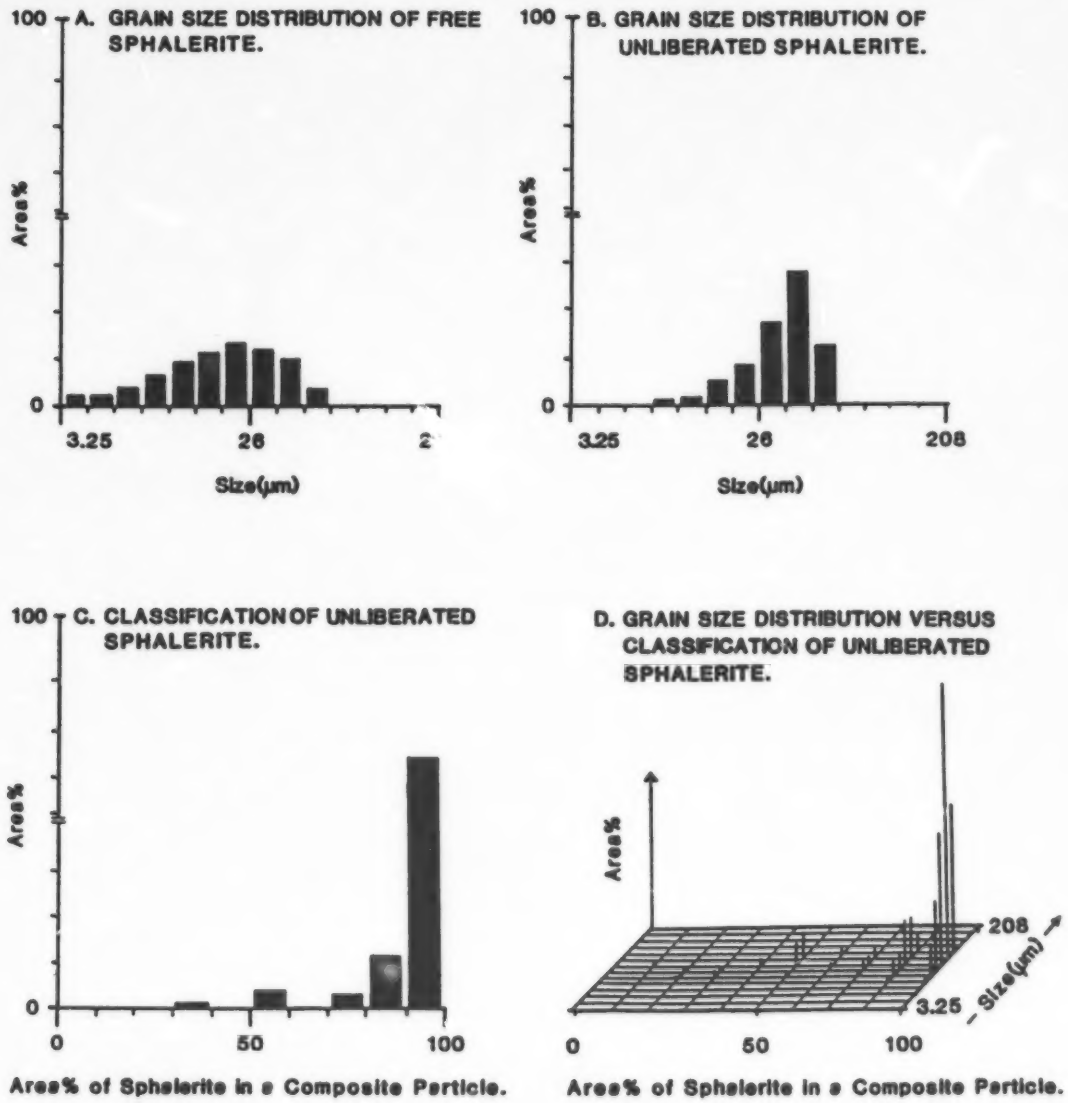

FIGURE 6.
PLOT OF THE RAW GRAIN SIZE AND LIBERATION DATA FOR CHALCOPYRITE IN THE Zn-CONCENTRATE.

Appendix E

TABLE 1

ELECTRON MICROPROBE ANALYSES OF SPHALERITE IN THE FLOAT FEED.

Wt %.	S.	Fe.	Cu.	Zn.	Cd.	Total.
1.	33.17	6.90	----	58.61	0.27	98.95
2.	33.12	7.42	----	58.92	0.15	99.61
3.	33.33	7.09	----	59.20	0.35	99.97
4.	33.19	7.93	0.08	57.76	0.43	99.38
5.	33.22	7.94	----	57.98	0.40	99.54
6.	33.20	8.07	----	57.99	0.39	99.66
7.	33.33	6.99	----	60.07	0.55	100.94
8.	33.21	6.91	----	60.10	0.11	100.32
9.	33.07	6.46	----	59.85	0.27	99.65
10.	33.18	7.05	----	58.72	0.23	99.18
11.	33.23	6.66	----	59.14	0.54	99.56
12.	33.11	2.66	----	64.87	0.48	101.11
13.	32.94	7.96	----	57.92	0.11	98.93
14.	33.17	7.93	----	57.14	0.24	98.48
15.	33.01	6.09	0.54	59.18	0.22	99.05
16.	33.15	7.04	----	59.16	0.14	99.49
17.	33.19	7.10	----	58.73	0.36	99.38
18.	33.34	6.87	----	60.46	0.21	100.88
19.	33.23	5.69	----	61.23	0.10	100.25
20.	32.87	7.00	----	59.58	0.20	99.65
21.	33.33	7.23	----	59.44	0.23	100.23
22.	33.24	6.43	----	59.38	0.29	99.33
23.	33.11	6.11	0.30	60.38	0.08	99.99
24.	33.40	7.32	----	59.79	0.27	100.77
25.	33.17	7.44	----	59.07	0.33	100.01
26.	33.18	6.96	----	60.14	0.12	100.40
27.	32.93	5.94	----	60.13	----	98.99
28.	33.00	6.48	0.43	59.93	0.28	100.12
29.	33.51	7.78	----	59.98	0.15	101.42
30.	33.28	7.11	----	60.11	0.37	100.87
31.	32.93	7.28	----	59.15	0.26	99.62
32.	32.91	7.76	----	58.17	0.47	99.31
33.	33.07	6.38	----	59.29	0.31	99.05
34.	32.87	6.74	0.56	58.53	0.40	99.10
35.	32.84	6.92	----	59.11	0.31	99.18

Wt %.	S.	Fe.	Cu.	Zn.	Cd.	Total.
36.	33.15	6.82	----	59.98	0.30	[illegible]
37.	32.87	8.19	----	57.23	0.06	[illegible]
38.	33.24	7.80	----	57.93	0.21	[illegible]
39.	32.84	6.68	----	58.89	0.25	[illegible]
40.	33.24	8.37	----	57.26	0.37	[illegible]
41.	33.30	7.16	0.16	59.18	----	99.80
42.	33.09	6.84	----	59.20	0.20	99.33
43.	33.35	7.46	----	59.45	0.18	100.44
44.	33.19	7.52	----	58.26	0.45	99.42
45.	33.40	7.53	----	58.82	0.45	100.20
46.	32.87	7.02	----	58.26	0.24	98.40
47.	33.02	6.87	----	58.63	0.26	98.78
X.	33.14	7.02	0.05	59.20	0.27	99.68
Max.	33.51	8.37	0.56	64.87	0.55	101.42
Min.	32.84	2.66	----	57.14	----	98.34

n = 47.

TABLE 2

ELECTRON MICROPROBE ANALYSES OF SPHALERITE IN THE Cu-CONCENTRATE.

Wt %.	S.	Fe.	Cu.	Zn.	Cd.	Total.
1.	32.87	6.74	0.44	60.49	0.20	100.74
2.	33.06	6.73	0.56	59.06	0.05	99.46
3.	32.80	6.57	0.58	59.67	0.23	99.83
4.	32.76	7.74	0.53	57.04	----	98.08
5.	32.86	7.17	0.54	59.23	----	99.80
6.	32.77	6.92	0.45	59.05	0.21	99.40
7.	32.89	6.43	0.82	59.63	0.29	100.03
8.	32.81	6.48	0.59	60.00	0.20	100.08
9.	33.16	7.67	0.65	59.73	0.07	101.29
10.	33.08	6.98	0.56	59.87	0.32	100.82
11.	33.07	8.07	----	58.24	0.65	100.02
12.	33.03	6.23	0.32	60.68	0.45	100.71
13.	33.43	7.40	0.54	59.89	0.40	101.66
14.	32.90	6.15	0.54	60.64	0.33	100.56
15.	33.11	8.05	0.77	58.00	0.15	100.08
16.	33.18	7.82	0.74	59.03	0.07	100.84
17.	32.86	6.91	0.65	59.29	0.27	99.98
18.	33.29	7.49	----	58.86	----	99.65
19.	33.00	6.39	0.56	60.52	0.36	100.83
20.	32.90	8.47	0.69	57.25	0.22	99.54
21.	32.99	7.43	0.64	58.79	0.16	100.02
22.	32.89	6.55	0.66	59.74	0.10	99.93
23.	33.15	7.35	----	59.05	0.49	100.04
24.	33.23	8.37	0.60	57.45	----	99.64
25.	33.73	7.32	0.63	58.04	0.24	98.96
26.	33.37	7.04	0.41	58.27	0.22	99.31
27.	33.10	7.02	----	58.70	0.19	99.01
28.	33.29	6.84	0.46	58.93	0.38	99.90
29.	33.28	6.84	0.44	59.93	----	100.49
30.	32.73	6.15	0.43	59.15	0.24	98.71
31.	32.87	5.91	0.72	59.54	0.31	99.34
32.	33.13	6.39	0.64	59.88	0.12	100.16
33.	32.87	6.20	0.38	59.15	0.28	98.88
34.	32.77	6.29	0.58	59.46	0.30	99.39
35.	33.15	4.96	0.51	61.25	0.28	100.16
36.	33.25	6.86	0.53	59.02	0.34	99.99
37.	33.17	7.09	0.52	59.00	0.23	100.11
38.	33.21	6.39	0.48	59.46	0.31	99.85
39.	33.42	6.74	----	59.91	0.22	100.29
40.	33.19	7.19	0.57	58.97	0.19	100.11

Wt %.	S.	Fe.	Cu.	Zn.	Cd.	Total.
41.	33.58	6.65	----	58.99	0.46	99.68
42.	33.32	6.86	----	59.91	0.21	100.30
43.	33.45	6.79	----	59.22	0.20	99.66
44.	33.16	7.78	0.94	58.54	0.12	100.54
45.	34.05	7.85	0.40	58.41	0.39	101.11
46.	33.19	5.82	0.46	60.68	0.09	100.24
47.	33.17	7.42	0.69	59.29	0.28	100.85
48.	33.26	5.98	0.64	60.16	0.34	100.39
49.	32.63	6.95	0.72	58.14	0.16	98.59
50.	32.90	6.97	0.44	60.03	0.44	100.78
51.	32.85	7.14	0.70	58.46	0.08	99.24
52.	33.04	5.71	0.79	60.85	0.16	100.55
$\bar{x}$.	33.07	6.91	0.49	59.28	0.23	99.98
Max.	34.05	8.47	0.94	61.25	0.65	101.66
Min.	32.63	4.96	----	57.04	----	98.08

n = 52.

TABLE 3

ELECTRON MICROPROBE ANALYSES OF SPHALERITE IN THE Cu-CONCENTRATE.

(+ 500 mesh fraction plus 20 % added carbon dispersant)

Wt %.	S.	Fe.	Cu.	Zn.	Cd.	Total.
1.	33.37	7.72	----	59.89	0.34	101.33
2.	33.24	7.81	----	59.46	0.22	100.73
3.	33.16	7.78	----	59.98	----	100.92
4.	33.42	7.21	0.45	59.08	0.05	100.20
5.	33.41	7.21	----	59.18	0.17	99.97
6.	33.45	7.70	----	58.38	0.24	99.77
7.	33.32	7.23	0.39	59.44	0.17	100.55
8.	33.33	3.06	0.62	64.33	0.27	101.62
9.	33.34	7.58	0.38	59.18	0.39	100.87
10.	33.39	7.44	0.50	58.36	0.27	99.96
11.	33.31	7.01	0.52	58.91	0.30	100.03
12.	32.84	2.95	0.65	64.52	----	100.96
13.	33.12	7.33	0.44	58.97	----	99.85
14.	33.24	7.76	----	58.38	0.34	99.73
15.	33.11	6.60	0.46	59.60	0.05	99.82
16.	33.19	7.21	0.39	59.71	0.09	100.60
17.	33.27	6.49	0.78	59.10	0.16	99.80
18.	33.07	3.83	0.54	62.11	0.24	99.78
19.	32.98	7.16	0.21	58.30	0.08	98.74
20.	33.24	7.34	0.27	58.68	0.37	99.91
21.	33.11	7.56	0.38	58.74	0.35	100.14
22.	33.24	7.30	0.77	58.33	0.12	99.76
23.	33.25	7.02	0.29	60.12	0.41	101.10
24.	33.24	6.60	0.43	59.17	----	99.43
25.	33.21	7.55	0.49	58.33	0.30	99.80
26.	32.99	5.65	----	60.87	0.28	99.79
$\bar{x}$.	33.23	6.77	0.35	59.66	0.20	100.20
Max.	33.45	7.81	0.78	64.52	0.41	101.62
Min.	32.84	2.95	----	58.30	----	98.74

n = 26.

Appendix F

GRAIN SIZE AND LIBERATION DATA FOR PYRITE IN THE Cu-CONCENTRATE.

TABLE 1

PARTICLE AND GRAIN SIZE DISTRIBUTIONS FOR PYRITE.

	All Minerals (Particles).	Total Pyrite.	Free Pyrite.	Locked Pyrite.
-3.25	0.21	0.00	0.00	0.00
-4.60	1.60	0.00	0.00	0.00
-6.50	4.02	3.28	4.21	2.55
-9.20	7.00	6.04	8.62	4.00
-13.0	9.78	10.1	12.9	7.89
-18.4	13.4	16.2	16.8	15.8
-26.0	18.4	19.4	20.9	18.2
-36.8	19.0	20.5	19.7	21.2
-52.0	15.3	20.4	12.4	26.7
-73.5	9.12	4.08	4.55	3.71
-104.	2.11	0.00	0.00	0.00

PYRITE = 184,000 um^2 : Free Pyrite = 81,500 um^2 i.e. 44.3 %.
Locked Pyrite = 104,500 um^2. i.e. 56.7 %.

Data collected using PRGM's 529, 530, 531 and 532, independently of all other image analysis data presented in this report.

TABLE 2

SURFACE AREA AND REL. AREA % OF LIBERATION CATEGORIES OF PYRITE.

Particles consisting of pyrite locked with specified mineral(s) only.	Surface Area. (um^2)	Rel. Area %. (um^2)
Gangue.	30,200.	16.4
Chalcopyrite.	28,700.	15.6
Sphalerite.	0.	0.0
Chalcopyrite + Sphalerite.	40,500.	22.0
Gangue (plus another).	38,400.	20.9
Chalcopyrite (plus another).	40,500.	22.0
Sphalerite (plus another).	6,690.	3.64

TABLE 3

CLASSIFICATION OF LIBERATION CATEGORIES OF PYRITE.

CLASS	Py-Gang. (only).	Py-Cpy. (only).	Py-Ternary (only).
0-10	0.12	2.37	7.32
10-20	0.12	0.45	3.86
20-30	0.00	2.56	2.55
30-40	1.71	2.12	4.00
40-50	1.34	2.69	9.41
50-60	3.84	4.62	2.59
60-70	5.73	2.24	8.50
70-80	8.54	14.0	4.68
80-90	25.4	25.3	35.3
90-100	52.9	43.5	21.7

TABLE 4

CLASSIFICATION OF LIBERATION CATEGORIES OF PYRITE.

CLASS	Py-Gang-Other (only).	Py-Cpy-Other (only).	Py-Sphal-Other (only).
0-10	8.08	7.34	5.43
10-33	7.22	6.99	13.8
33-55	14.5	13.9	1.08
55-78	7.25	12.1	31.9
78-100	63.3	59.9	47.8

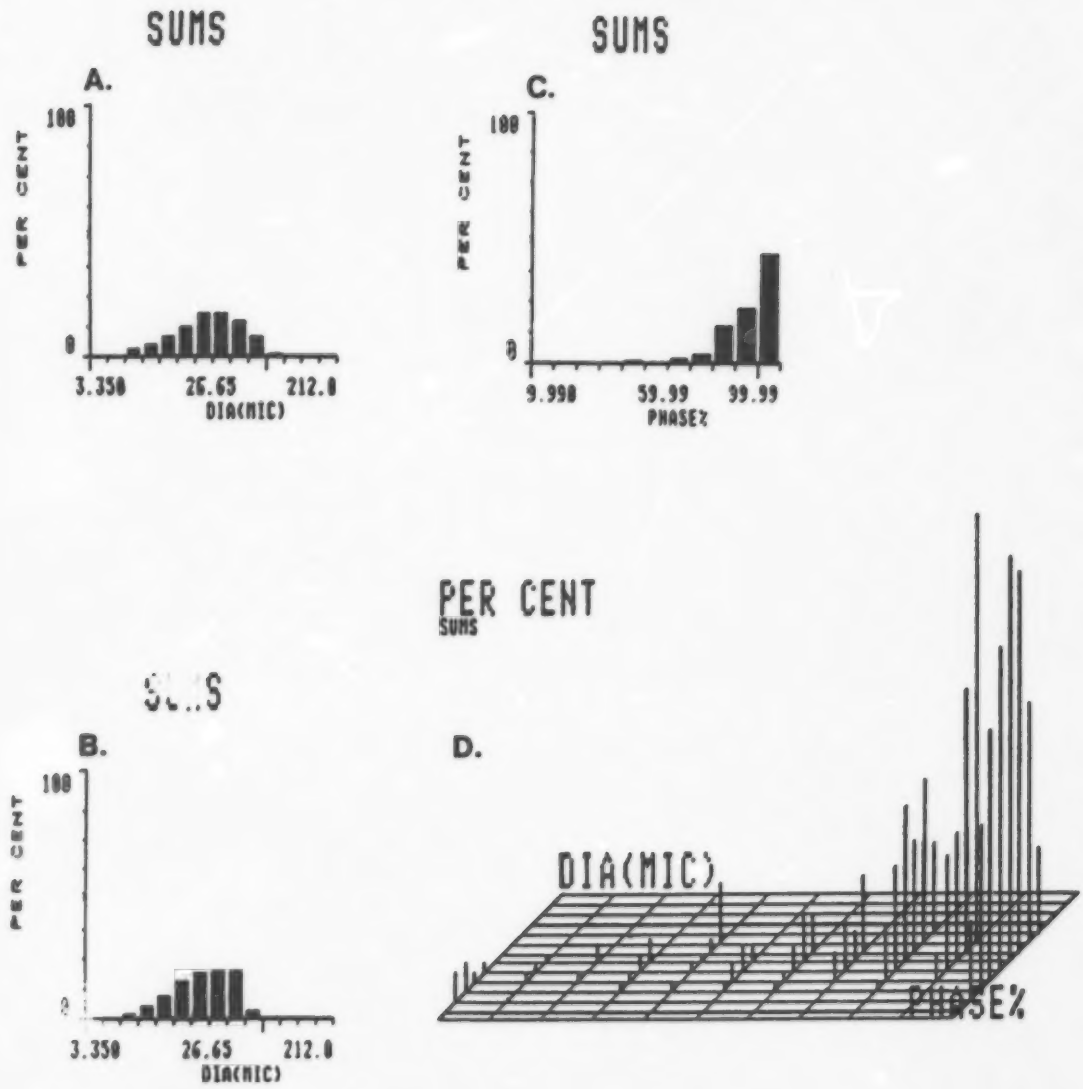

FIGURE 1.
PLOT OF GRAIN SIZE DISTRIBUTIONS OF (A) ALL MINERALS (I.E. PARTICLE SIZES), AND (B) PYRITE IN THE Cu-CONCENTRATE. THE CLASSIFICATION OF UNLIBERATED PYRITE (PLUS FREE) IN THE Cu-CONCENTRATE IS SHOWN IN (C), AND PLOTTED AGAINST GRAIN SIZE IN THE 3-D DIAGRAM (D).